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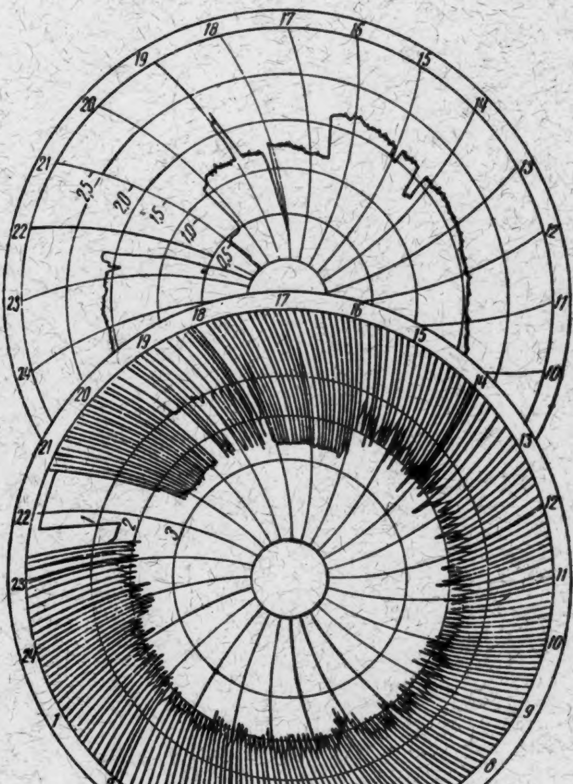

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THE MAIN IRON ORE DEPOSITS OF THE USSR

I. S. Shapiro

Soviet geologists have performed an enormous task in determining the regularities in distribution and the form of occurrence of iron ores, and they have calculated the iron ore reserves.

As a result of the work done, known iron ore reserves of the country increased rapidly. Whereas, the iron ore deposits of tsarist Russia were estimated to be 2 billion tons, the iron ore deposits so far explored in the USSR exceed 37 billion tons, constituting about 40% of the world's iron-ore resources.

The table gives the distribution of available reserves of iron ores of the USSR, according to economic regions and major deposits (January, 1957).

Iron Ore Reserves of the USSR Regions and major deposits	Reserves of commercial grade ores on January 1, 1957	
	million tons	%
North-West Regions	1681.6	4.8
Central Regions	7251.6	20.5
out of this quantity: Deposits of the Kursk Magnetic Anomaly Region	4061.3	11.5
Southern Regions	10,819.7	30.6
out of this quantity: Krivoi Rog deposits	8331.3	23.6
Kerch deposits	1654.3	4.7
North Caucasus and Transcaucasus Regions	108.5	0.3
Total, Western Regions	19,861.4	56.2
Urals Regions	5800.4	16.4
Mount Magnitnaya deposit	282.6	1.0
West and East Siberia and Far East Regions	3365.6	9.6
Kazakhstan Regions	6286.0	17.8
Total, Eastern Regions	15,452.0	43.8
Total, USSR	35,313.4	100.0

Out of the total of explored supplies only 15% consist of rich ores which contain 55-57% iron and require no beneficiation; these are the rich Krivoi Rog deposits containing hematite, martite and magnetite, the Kursk deposits containing siderite, martite and hematite, a part of the magnetite ores in the Urals and Kazakhstan, and some other deposits.

Most of the explored supplies of commercial deposits contain ores which require beneficiation:

1) the magnetite ores of the Urals, West and East Siberia, and Kazakhstan (average iron content 40%);

2) the ferrous quartzite ores of Krivoi Rog, Kursk, North-West Region and the Far East (average iron content 30-37%);

3) the embedded-titanium magnetite ores of the Kachkanar deposit (iron content about 17%);

4) the brown hematites of Kerch, Ayat, Lisakovskoye and other deposits (average iron content about 37%);

5) the hematite ores of the Angara-Pit deposits (average iron content 40%).

Approximately one third of all the iron ore supplies consist of ores whose beneficiation is difficult; these are hematite quartzites of Krivoi Rog, brown ores of Kerch, brown hematites of Ayat and Lisakovskoye and some other ores which require complex methods of beneficiation, involving gravitational concentration, flotation and magnetic roasting.

North-West, West and North Regions. The major deposits of the North-West Economic Region and those of Olenegorsk and Ena-Kovda which provide raw materials for the Cherepovets Metallurgical Works.

The Olenegorsk deposit consists of magnetite quartzites. It is estimated at 337.2 million tons, and the average iron content in the ore is 32.5%. Since 1955, the ore has been open-worked. The ore is beneficiated by the gravitational and magnetic methods and the concentrate, containing 58-61% iron, is supplied to the Cherepovets Works. A mine is under construction at the Ena-Kovda deposit which will also supply the Cherepovets Works with ore.

Central Regions. Here is a rich iron ore district, known as the Kursk Magnetic Anomaly, with inexhaustible supplies of rich ores and magnetic quartzites. The iron ore deposits of the Belgorod Region, unique in their size and quality, are composed of rich hematite-martite and siderite-martite ores, as are the smaller deposits of Mikhailovka, Lebedin and other deposits of the Kursk Magnetic Anomaly.

The iron ore deposits of the Belgorod Region are estimated at 10-12 million tons of rich ore, containing about 60% iron. The ores, however, are situated in difficult hydrogeological conditions at a depth of over 450-500 m. The ores of the Mikhailovka and Lebedin districts are situated much more favorably and are suitable for open pit mining. New mines in those districts are now under construction. A mine and a beneficiation plant are also being constructed in the Iuzhno-Korobkovo district of ferrous quartzites containing 33% iron.

So far, in the Central Regions, only the brown hematites of the Tula and the Lipetsk iron ore deposits are exploited and an experimental mine for the ferrous quartzites of the Kursk Magnetic Anomaly is in operation.

South Regions. The main iron ore regions of the Ukraine are the Krivoi Rog and the Kerch deposits.

The Krivoi Rog iron ore deposits — the main iron ore supplies of the iron and steel industry of the South — provided over 44% of the total iron ore output of the USSR.

The Krivoi Rog iron ores are divided into rich ores and ferrous quartzites. The rich ores are mined mainly in deep underground mines.

The exploitation of ferrous quartzites (containing 33-39% iron) assumes more and more importance. In 1955, the South Ore Beneficiation Combine, with an annual output of 4.5 million tons of concentrate from magnetite quartzites, was put into operation in the Krivoi Rog Region.

The Kerch ore consists, in the main, of brown hematites. The average composition of ores in the Kerch district is: 28-29% Fe, 0.8-4.3% Mn, 0.6-1.1% P, and 0.07-0.13% As. The high percentage of phosphorous makes it possible to utilize the ores as combined raw materials. The Kerch district supplies the ore to the "Azovstal" Works. Because of the high phosphorus content and the presence of arsenic in the Kerch ore concentrates, up to 25% by weight of Krivoi Rog ore is introduced into the blast-furnace charge at the "Azovstal" Works. The phosphorus slag, obtained during the processing of the Kerch pig iron in open-hearth furnaces, constitutes a valuable fertilizer for agriculture.

North Caucasus and Transcaucasus Regions. The Dashkesan ore deposits whose ore goes to the Zakavkaz Metallurgical Works in Rustavi, are estimated at 86.2 million tons, the average iron content being 30%.

The unexploited Malka deposits in Northern Caucasus, which are estimated at 14.8 million tons and contain 32.3% iron, may be mentioned. The ore from these deposits requires complicated, expensive and uneconomical methods of beneficiation.

Regions in the Urals. The ore mined in the Urals constitutes 40% of the total ore output of the USSR. The iron ore deposits in the Urals are in nine mining regions: Ivdel, Bogoslovka, Tagil-Kushva, Kachkanar (68% of

the total ore reserves), Alapaevsk, Bakal, Ligazinskii-Komarovsk, Magnitogorsk and Orsk-Khalilovo Regions.

The Urals deposits occupy third place in the country with regard to the amount of explored ores but only 7% of the Urals ore can be used without beneficiation.

A large amount of ore is mined from the Magnitogorsk deposits, containing sulfide magnetite varieties (primary ores) with sulfur content of 1%-2% or more, and oxidized martite ores. Rich oxidized ores containing, one the average, 58.7% iron, are subjected only to crushing before being smelted, and the fines from the crushing operations are sintered. Lean oxidized ores (38% Fe) are beneficiated on jigging machines and the fine fractions of the concentrate are sintered. The sulfide ores, even if rich in iron (up to 55%), are finely ground, subjected to magnetic separation and sintering in order to remove sulfur.

The Magnitogorsk deposits are at present supplying the Kuznetsk Metallurgical Combine with ore. In the near future, however, in connection with the envisaged expansion of the Magnitogorsk Metallurgical Combine, the Magnitogorsk supplies will not be adequate to satisfy even the Magnitogorsk Works,let alone other works, and by 1980, the Magnitogorsk ore deposits will be exhausted. The problem of ore supplies for the Magnitogorsk Combine is being satisfactorily solved by the development of an iron ore mining district in the Kustanai Region, at first as an auxiliary and later as the main iron ore supply base for the Combine.

The deposits of the Bakal district are represented by brown hematites of 45% average iron content and high manganese content, and by siderites of 35% average iron content. The Chelyabinsk, the Satka and the Ashinskii Metallurgical Works are supplied with these ores and in future they are to be supplied by the Kustanai magnetite ore.

There are several deposits of magnetite ores (Mount Blagodat, Mount Visokaya, Lebiazhe, Estiushinsk and some others) in the iron ore district Tagil-Kushva. The ore from these deposits is supplied to the Nizhne-Tagil Metallurgical Combine and to the Kushva and the Nizhne-Salda Metallurgical Works.

The Kachkanar district is comprised of two deposits: the Gusevogorskoe and the Kachkanar deposits. Explored supplies are estimated at 3.9 billion tons of titanium-magnetite ores containing 16-17% iron as well as some titanium and vanadium. So far, these deposits have not been mined, but now plans are being prepared for the utilization of Kachkanar ores, since it has been found that they are easily beneficiated by the method of wet electromagnetic separation. From the Kachkanar ore concentrates a good quality sinter is obtainable. This deposit will become a subsidiary ore supply base for the Nizhne-Tagil Combine.

The Orsk-Khalilovo group of deposits of naturally alloyed iron ores containing chromium, nickel and cobalt, is comprised of the Akkerman, Novo-Kiev, Novo-Petropavlovsk and some other deposits which constitute the iron ore base for the Orsk-Khalilovo Metallurgical Combine. Only the Novo-Kiev deposit is mined at present.

Kazakhstan's Regions. The explored ore deposits of Kazakhstan constitute over 6 billion tons, i.e., over 20% of total ore resources of the USSR. About 83% of Kazakhstan's ore supplies are in the Kustanai Province. These are mainly magnetites and brown hematites.

The largest magnetite deposits, containing 46-48% iron on the average, are the Sokolovka, the Sarbai and the Kachkanar Districts in the Kustanai Province. Almost all the ores of these districts have high-sulfur and low-phosphorus content. The ores require magnetic separation and sintering. The explored supplies of magnetite ores of the Kustanai Province constitute more than 2 billion tons.

The Sokolovka and the Sarbai deposits are to form the raw material base for the Sokolovka-Sarbai Mining and Concentration Works, which is now under construction and which will have a capacity of 15 million tons of crude iron ore per year. In September 1957, the Works supplied its first delivery of ore for the Urals works.

Still larger supplies of iron ore have been explored in the districts with brown hematite deposits. In particular, the Ayat and the Lisakovskoye deposits comprise more than 3.6 billion tons of ore. These deposits are hardly inferior to the world famous Lotharingia (Lorraine) deposits in Western Europe which constitute the basis of the iron and steel industry of West Germany, France and Belgium. The mining of the Ayat and the Lisakovskoye deposits is easy, since in several places the ore is found on the surface. The ores contain phosphorus and vanadium. Their beneficiation requires magnetising roasting. Pig iron from these ores has to be processed by the

Thomas method. Phosphorus-containing slags from this process constitute a valuable material for agriculture.

Three groups of deposits are known in the Karaganda Province. These are: Atasu ky, Karsakpai and Ken-Tiube-Togai. The Atasusky group of rich hematite ores is comprised of several iron ore deposits, of which the West Karajal deposit is mined; a new mine is planned for the Bolshoi Ktai deposit. These deposits will provide ore supplies for the Karaganda Works, now under construction.

West Siberia Regions. The explored iron ore deposits of West Siberia, estimated at 265 million tons, are concentrated on the Kemerovo Province and the Altai Territory. By far the largest part of these deposits is in the Gornaia Shoria iron ore and coal district which constitutes the raw-material base of the Kuznetsk Metallurgical Combine.

The deposits of Gornaia Shoria are mostly magnetite ores and, to a lesser extent, martite and semimartite ores with 40-42% iron content; most of the ores contain sulfur and very frequently have zinc inclusions, some of the ores contain cobalt. Quality concentrates are obtained from these ores after the beneficiation by means of electromagnetic separation, but some ores can be smelted without prior beneficiation.

In 1955-1957, enormous deposits of sedimentary origin iron ore, estimated at many billion tons, have been discovered in the Tomsk Province. This is the first West-Siberian iron ore base. The most promising is the Bakchar district. Prospecting and exploration work in this region is still in progress. The region holds very considerable promise for the future.

The Regions of the Krasnoyarsk Territory and the Tuva Autonomous Province. Two large iron ore districts have been discovered here – the Khakassia and the Angara-Pit deposits.

The most promising are easily beneficiated magnetite ores of Khakassia, and in particular, the Abakan and the Teisk deposits. The Abakan deposits, situated 176 km southwest of the town of Abakan, constitute 8.3 million tons and contain 47% iron on the average, the majority being rich ores of 50% or more iron content. The explored supplies of the Teisk deposit are estimated at 117 million tons, the average iron content being 34.6%. Both deposits constitute part of the raw material base of the Kuznetsk Metallurgical Combine. The largest deposits of the Angara-Pit district are the Nizhne-Angara and the Ishimbinskoye deposits, containing 40.5% iron, 0.02-0.03% sulfur and 0.04-0.07% phosphorus. The ores require complex and expensive beneficiation methods, including flotation and magnetic roasting.

The West Siberia Regions. Explored ores of West Siberia are estimated at 1300 million tons. The Angara-Ilim Region, the largest iron ore region in West Siberia, accounts for about 50% of all the explored ore supplies of West Siberia. The Korshunovo and the Rudnogorskoye deposits are the largest in that Region.

The magnetite ores of the Korshunovo deposit contain 33.7% of iron and in their properties are similar to self-fluxing ores. They are satisfactorily beneficiated by wet electromagnetic separation. Owing to large supplies of magnetite ores, the ease of their beneficiation, a high quality of the sinter and favorable conditions for mining and transportation, the Korshunovo deposit is rapidly being developed. A mine has been planned and its construction is about to be started.

The explored supplies of the Rudnogorskoye deposit are estimated at 208 million tons of magnetite ores containing 46% iron on the average. From the point of view of the technology of processing, the ore of the Rudnogorskoye deposit is similar to the Korshunovo ore.

Large deposits of iron ore have also been discovered in Yakutia and the Chita Province.

The South-Aldan Iron Ore district, situated in the south part of the Yakutsk ASSR, consists of several deposits. The best explored are the Taezhnyi, Pioner and Sivaglinskoye deposits. The ore supplies of the South Aldan iron ore district are estimated at 1.4 million tons. In the Chita Province, the large Berezovskoye deposit of brown hematites, containing 39% of iron on the average, has been explored.

Far East Regions. The largest iron ore deposits of the Far East are the Kimkan and the Garinskoe deposits. The first one is estimated at 190 million tons. The ores consist of magnetite quartzites of 35.5% average iron content and a fairly high phosphorus content (about 0.2%).

The Garinskoe iron ore deposit is situated in the Mazanovskii Region of the Amur Province. Its reserves constitute 160 million tons of ore. The ores are magnetites of 46.9% average iron content.

THE BLAST FURNACE INDUSTRY

THE TECHNOLOGICAL REGIME OF THE OPERATION OF BLAST FURNACES AT THE KMK

Cand. Tech. Sci. N. N. Chernov

The Siberian Metallurgical Institute

Operational parameters for a smooth running of each furnace and which ensure a maximum output at an economic input of raw materials and fuel have been selected and tested for each furnace of the blast furnace plant at the Kuznetsk Metallurgical Combine. After the blast furnace plant has been modernized, these parameters are almost identical for all furnaces,if the slight variations in the temperature of hot blast, the changes in the ratio of ore to sinter in the charge, and the slight fluctuations in the amount of blast are ignored.

Most of the time, the blast furnaces of the KMK (Kuznetsk Metallurgical Combine) are producing low-manganese conversion pig iron of the following composition (average figures for 1956), %:

Si	Mn	S	P
0.76	0.53	0.049	0.15

The approximate composition of the charge is: sinter 75%, Magnitogorsk ore 1.0-2.0% and Tashtagol ore 23-24%. The basicity of the sinter is 1.4 at an average iron content of 51-52%. The input of crude limestone constitutes 55-60 kg per ton of pig iron. All blast furnaces are operated at 0.6-0.65 atms gas pressure at the furnace top. The pressure drop between the bottom and the top of the furnace does not usually exceed 1.26-1.3 atms. Coke consumption for five months of 1958 constituted 670 kg per 1 ton of pig iron and the coefficient of furnace volume utilization (the ratio of blast furnace useful volume to daily production) was 0.667.

Until 1954, the furnaces produced ordinary conversion pig iron of the following composition (average for 1950), %:

Si	Mn	S	P
0.75	1.86	0.04	0.27

For 8-10 days once every quarter, foundry pig iron is produced in the furnaces. As the production of low-manganese conversion pig iron has been maintained for a long time, it became possible to standardize the regime of furnace operation for this process.

The blast regime. The volume of the blast entering the furnace per one minute is approximately twice the useful volume of the furnace.

It is well known that the output of the furnace is directly proportional to the amount of the blast. Therefore, the furnace operators always aim at maintaining a full-rate blast. A reduction in the blast is resorted to only in extreme cases, and as soon as the furnace operation is corrected, the blast is brought back to normal.

It is however, uneconomic to increase the blast rate above the established optimum limit. The blast regime must conform to the gas permeability of the stock, and the blast rate can be increased above the optimum limit only when the charge has been improved with respect to its sieve analysis. Otherwise, an increase in furnace output will not be accompanied by a decrease in coke consumption, since, as the result of an excessive increase in blast rate,the efficiency of the reactions between the gases and the charge declines and hence the coke consumption increase.

As a rule, prolonged excessive blasts lead to the deformation of the furnace crosssection and finally result in decreased output. Therefore, the blast-furnace operators at the KMK always establish the blast regime in full conformity with the requirements of the materials employed (Fig. 1).

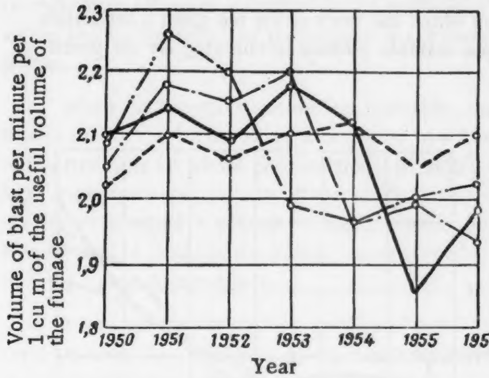


Fig. 1. Ratio of the rate of blast per minute to the useful volume of the furnace at the KMK. ——— Furnace No. 1; - - - - - - furnace No. 2; - · - · - · - furnace No. 3; -·-·-·- furnace No. 4.

When the gas permeability of the stock is improved, the furnace foreman slowly and cautiously increases the blast to a certain limit; when the limit is exceeded, slips of the stock occur and signs of channeling appear. In this way, the foreman determines the equilibrium conditions between the rate of blast and the regular operation of the furnace, and in the course of his work he adheres to those conditions. The amount of air entering the furnace is strictly controlled by the gas watchman and the foreman. Every hour, the actual air input is evaluated and compared with the assigned input. For this purpose, there are special slide rules for each furnace at the KMK, by means of which the required air input and the corrections for hot blast pressure, cold blast temperature and blast humidity can be calculated.

It should be stressed that each furnace is provided with a predetermined value of air volume per minute which is required for a maximum blast furnace output and a minimum cost of pig iron. Every blast furnace worker should be able to calculate this volume. At a constant quantity of raw materials and coke, the air rate depends on the cross section of the furnace.

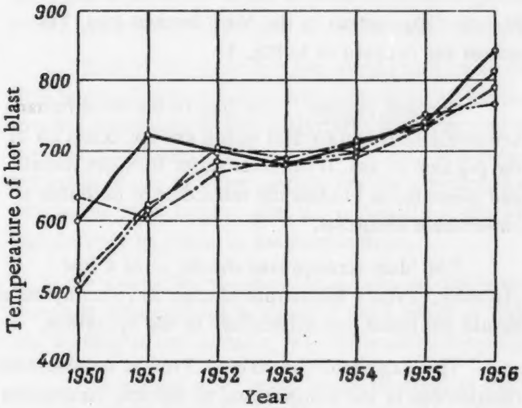


Fig. 2. The temperature of the hot blast in the furnaces at the KMK. The curves are denoted as in Fig. 1.

The temperature regime. At the blast furnace plant of the KMK, a temperature of 820-850°C is accepted as the normal temperature of the hot blast. The blast at that temperature can be ensured for a very long time by the air stoves. The moisture content of air is 25 g/cu m. When recalculated on dry air basis, the temperature becomes 625-675°C.

The temperature and the rate of blast are closely related and depend on several factors: the quality of the charge materials (mainly their particle size), the distribution of the materials in the top of the furnace, thermal conditions and the condition of the internal walls of the furnace. If the blast and the temperature regimes do not conform with these conditions, the regular operation of the furnace will be upset. The blast furnace operators at the Kuznetsk Combine consider that the dependence of the volume of gases which pass through the stock on the amount of free passages in the stock is the controlling factor in the majority of cases of furnace irregularities connected with the temperature and the rate of the blast.

In each specific case the foreman must decide how to make the best use of an improved gas permeability of the charge: whether to intensify the blast-furnace process by increasing the rate of the blast or by increasing the temperature.

At the Kuznetsk Metallurgical Combine, the foremen consider it preferable to increase the temperature of the hot blast (Fig. 2). The heat of the blast should be absorbed within the hearth and the bosh space. If, however, this heat is utilized for increasing the temperature of combustion products, the velocity of blast furnace gases and their thrust increase. Hence, the smooth operation of the furnace will be upset. In order to prevent this, the blast

furnace foremen at the KMK, when raising the temperature of the blast, increase the ore burden per ton of coke.

It can be said without exaggeration that the main trend in the operation of the blast furnace plant at the KMK is the tendency to increase the ore charge per ton of coke (Fig. 3). The increase in the ore burden promotes a better utilization of the chemical energy of the gas stream (Fig. 4). However, when increasing the ore portion of the charge one must take care not to exceed the limit above which the stock offers too great a resistance to the passing gases. When the ore burden is increased, the most suitable manner of charging for the given conditions should be selected.

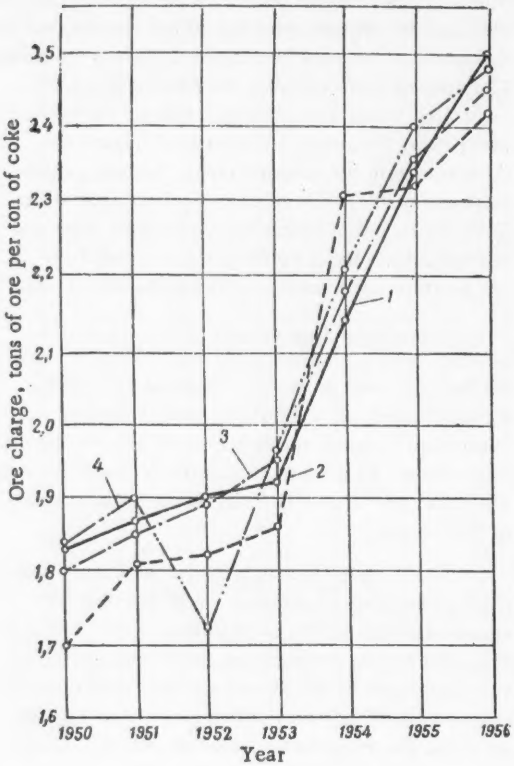


Fig. 3. The ore burden in the furnaces of the KMK. The curves are denoted as in Fig. 1.

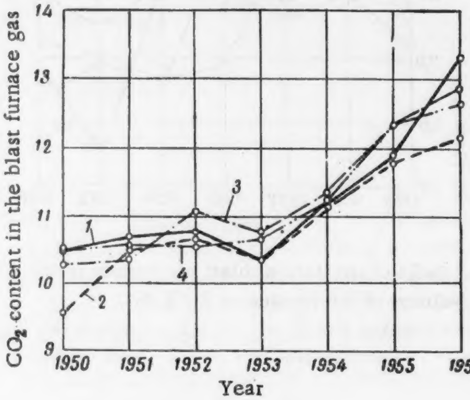


Fig. 4. CO_2 content in the blast furnace gas. The curves are denoted as in Fig. 1.

The slag regime. The slag in the blast furnace accumulates the materials which are not taken up by the pig iron or gas, it removes sulfur from the metal and promotes or hinders the reduction of desirable or undesirable elements.

The blast furnace slag should be of a low viscosity. Even a substantial change in its composition should not cause any difficulties in the operation.

The slag should be stable. Even at considerable fluctuations in the composition of the slag its desulfurizing power should not change markedly. Otherwise, the foreman has to reduce the ore burden in order to increase the heat content, and this would lead to a fall in the output of pig iron and a rise in coke consumption.

During the production of conversion pig iron of an increased manganese content, at the KMK, the slag had the following composition, %:

SiO_2	Al_2O_3	CaO	MgO	MnO	FeO
38.7	13.1	37.9	5.0	3.2	0.65

The basicity of the slag was 0.95-0.97 and its yield amounted to 0.6-0.65 tons per ton of pig iron. The slag had a satisfactory desulfurizing power.

During the production of low-manganese pig iron, the slag had the following composition, %:

SiO_2	Al_2O_3	CaO	MgO	MnO	FeO
37.5	13.45	40.7	5.4	0.9	0.65

The elimination of the lean Mazulskii manganese ore from the charge caused a fall in the yield of slag to 0.6-0.56 tons per ton of pig iron. It also resulted in a decrease in the manganese content in the slag.

In order to prevent an increase in the sulfur content in the pig iron caused by the fall in the yield of slag and by the decrease of the desulfurizing action of manganese, it became necessary to increase the basicity of the slag to 1.05-1.1. A reduction in the content of manganous oxide in the slag does not affect the working of the furnace, but the final slags are more viscous and less stable and hence make work at the hearth more difficult. In order that a regular operation of the furnace be maintained, such slags should be well heated. The most stable regular operation of the furnace is achieved at a well heated hearth, when the pig iron contains 0.8-0.9% silicon.

When the hearth is heated less intensely, the slags of low manganous oxide content do not flow down the hearth easily. If, at the same time, coke of a lower quality is used, the hearth becomes partly blocked and is not active over its whole cross section. In such a case, the furnace operators at the KMK "flush" the hearth with high manganese pig iron and slage which have a high manganous oxide content. Sometimes, the manganese ore in the charge is replaced by hearth cinders. Some foremen charge the hearth cinders in small batches but this method of flushing the hearth is less effective and more prolonged, although, in this case, the temperature of the furnace does not fall.

When the hearth cinders instead of a full round of the ore are charged, the temperature of the hearth must be taken into account. If the temperature of the hearth is low, a blank charge should be allowed or a few skips of additional coke should be charged.

At the KMK, after 1 to 2 months of the operation of the blast furnace with low manganese pig iron, the change to the production of pig iron of a higher manganese content (up to 1.0%) is effected in the course of 1-2 days.

The charging regime. Usually the manner of filling the furnaces is coke-ore-ore-coke-coke-stone or ore-ore-coke-coke-coke-stone, alternating after a predetermined number of rounds with coke-ore-ore-coke-coke-stone. An additional manner of charging is coke-coke-ore-stone-coke-coke-ore-stone.

There is the view that one should aim at an "ore first" method of charging as the most economic one. This is not always correct. When deciding on the manner of charging, one must take into consideration the specific conditions of raw materials and the actual cross section of the furnace. Sometimes, the "ore first" method of charging the furnace may prove less economic than the "coke first" method. The criteria for the selection of one or other method are the consumption of coke and the output of the furnace. The content of CO_2 in the blast furnace gas, the regularity of the working of the furnace and the constancy of the thermal regime may be taken as indirect indices.

As a rule, the foremen do not change the manner of filling at the same time when any other measure, aimed at adjusting the working of the furnace is undertaken, since, with a large number of factors affecting the working of the furnace, it is difficult to determine the effect of each factor separately.

The normal stock level in the furnaces at the KMK is 2 m. In each actual case, the stock line is predetermined depending on the particle size of the charge materials. Thus, if the amount of the Tashtagol coarse ore in the ore charge is increased, the stock level is lowered; and if the amount of sinter or the amount of Magnitogorsk ore is increased, the level is raised. Therefore, the changes in the size composition of the ore do not affect the gas flow.

The ore occupies a large area of the cross section of the furnace top when the stock level is raised and, vice versa, it accumulates at the walls when the level is lowered because, when the level is raised, the depth of the cone increases and when it is lowered the cone depth decreases. Approximate variations in the stock level constitutes 0.2-0.4 m.

The usual weight of the coke charge at the KMK is 5.0-5.5 t. Under normal conditions, the revolving distributor operates in all six positions (at furnaces No. 2 and No. 4 – in eight positions).

Recently, the coke-ore-coke-ore-coke-stone manner of filling the furnace has been adopted at the blast furnace plant. To prevent the ore in a given charge from falling always on one side, the change in the position of the revolving distributor is effected, not at the end of the charge, but after the first ore skip.

The temperature of the furnace is controlled by the weight of the coke charge.

THE UTILIZATION OF MANGANESE IN FERROMANGANESE DURING THE OPERATION OF THE BLAST FURNACE WITH OXYGEN-ENRICHED BLAST

A. N. Red'ko

Head of the Central Works Laboratory of the Novo-Tula Metallurgical Works

Oxygen enriched blast has been employed in blast furnace No. 1 at the Novo-Tula Metallurgical Works since November, 1948. In April, 1951, the production of ferromanganese was started in that furnace.

The purpose of the study of ferromanganese production in furnace No. 1 was to attain the maximum productivity of the blast furnace and, therefore, one was aiming at a high intensity of coke combustion for all the various kinds of charge. The first period of the production of ferromanganese in this furnace extended from April to November 1951. Since August, 1955, and up to the present day, the furnace produces ferromanganese from Chiatura ore. The manganese content in the ore was 36-45%. Coke of 1.6-1.8% sulfur, 10-11% ash and 2-7% moisture content, supplied by the Moscow Coke and Gas Works was employed. The oxygen content in the blast during the production of ferromanganese constituted 30-33%. Mean indices of the operation of blast furnace No. 1 with oxygen-enriched blast and various kinds of the charge, are given in the table below.

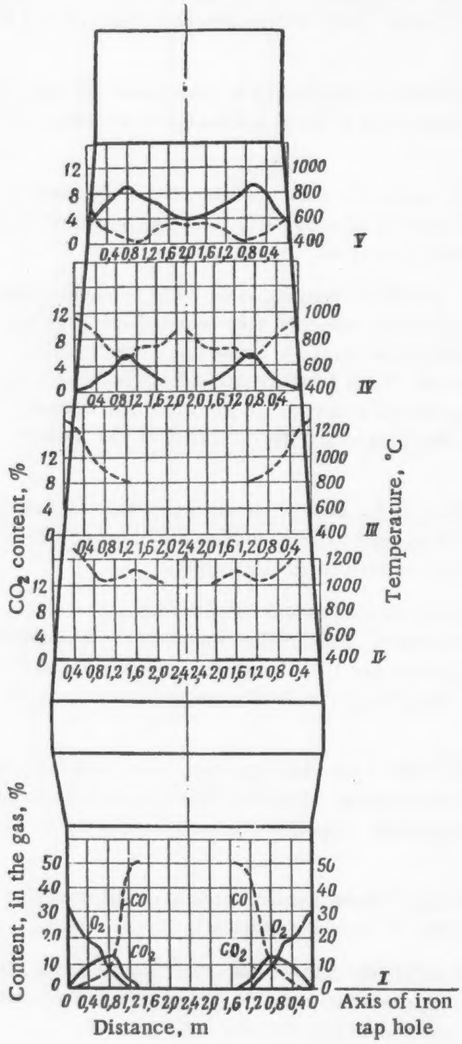


Fig. 1. Gas composition and temperature at various levels in the furnace: continuous line represents CO_2 content; dotted line represents temperature.

It is seen from the table that when the furnace is operated with oxygen-enriched blast, the furnace process is accelerated to a high degree. The rate of coke combustion on the production of ferromanganese in furnace No. 1 varied within the limits of 1.6 to 1.3 tons of coke per 1 cu m of furnace volume per 24 hours, depending on the quality of the charge and the condition of the furnace.

The weight of coke charge varied within the limits of 2.8-3.2 tons. The furnace was charged mainly according to the following methods: 5 ore-ore-coke-coke-stone – 5 coke-coke-ore-ore-stone, 4 ore-ore-coke-coke-stone – 3 coke-coke-ore-ore-stone or ore-ore-coke-coke-stone. The stock level was varied within 1.5-3.0 m. To reduce the temperature of the top, the coke was sprayed with water (0.25-0.5 cu m per charge). Curves representing the gas composition and temperature at various levels in the furnace are shown in Fig. 1.

Owing to the variations in the quality of the charge, the large input of limestone and the high yield of slag, there was a tendency towards acidic slag formation. Hence, the basicity of the slag varied considerably in different periods of furnace operation. According to monthly reports, the transfer of manganese into pig iron amounted to 77.8-66.0% (Fig. 2).

The main losses of manganese occur in the blast furnace dust and the slag. The manganese losses in blast furnace dust, scrap and gases constituted 10-25% during the period considered. In 1956, the manganese loss in the blast furnace dust and gases was 11.5% on

Data on the Operation of Blast Furnace No. 1 of the Novo-Tula Metallurgical Works During the Production of Ferromanganese

Indices	Periods		
	September-November, 1951	September, 1955-February, 1956	March-May, 1957
Coefficient of useful volume utilization (ratio of blast furnace useful volume to daily output)	0.428	0.5	0.57
Oxygen content in the blast, %	30.6	33.2	31.9
Rate of coke combustion, t/cu m/24 hrs	1.6	1.4	1.33
Blast			
temperature, °C	812	812	876
pressure, atms	1.03	0.94	0.81
Blast furnace gas:			
temperature, °C	215	220	315
pressure, atms	0.27	0.1	0.085
CO_2 content, %	7.3	7.7	7.5
Input per ton of pig iron:			
dry coke, tons	1.673	1.727	1.805
1st and 2nd grade, washed manganese ores, tons	2.246	2.076	1.047
ordinary manganese ore, tons:			
1st grade	–	0.229	0.839
2nd grade	–	0.173	0.501
metallic additions, tons	0.163	0.204	0.225
fluxes, tons	0.594	0.695	0.897
oxygen, cu m	458.2	600.0	599.0
Manganese content, %:			
in ore portion of the charge	47.8	46.3	43.25
in pig iron	75.4	73.9	72.85
Slag:			
basicity, $\frac{CaO}{SiO_2}$	1.26	1.3	1.35
$\frac{CaO + MgO}{SiO_2}$	–	–	1.43
yield per ton of pig iron, t	0.836	0.945	1.144
MnO content, %	19.25	15.5	12.89
Dust carry-out per ton of pig iron, t	0.13	0.238	0.114
Distribution of manganese charged into the furnace, %:			
in pig iron	42.9	69.2	77.5
in slag	12.2	10.7	12.1
in scrap	2.3	3.67	1.5
in collected dust	5.5	7.26	3.4
siftings and losses in the gas	7.1	9.17	5.5
Number of settlings per 24 hrs	2.6	0.8	0.96

the average. These losses can be reduced by a steadier working of the furnace at a lower rate of the blast furnace process, or by means of sintering the pulverulent manganese ores.

The losses of manganese in the slag constituted 10.0-15.6% (average losses in 1956 were 13%). These losses can be reduced by decreasing the yield of slag, by increasing its basicity and by increasing the temperature of the hearth. Under the conditions which obtain at the Novo-Tula Works, a reduction in the manganese losses in the slag can be achieved only if the blast furnace process is conducted at a high temperature of the metal and a high slag basicity.

Figure 3 shows the dependence of the transfer of manganese into pig iron on the basicity of the slag; with the increase in the basicity, a more efficient transfer of manganese into pig iron took place.

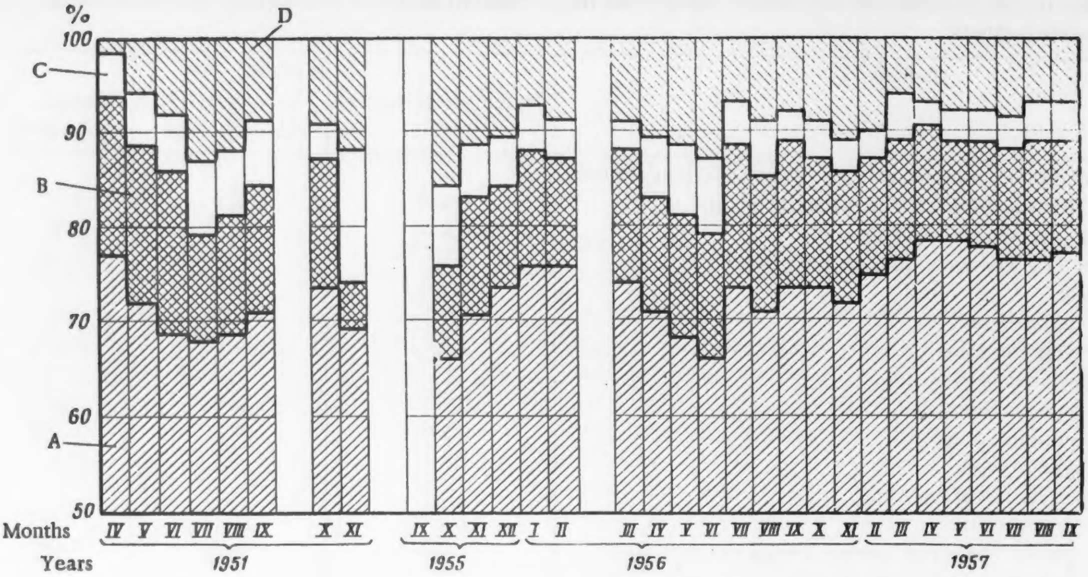


Fig. 2. Transfer of manganese from ore into pig iron:
A) transfer into pig iron; B) losses in slag; C) losses in dust; D) losses unaccounted for.

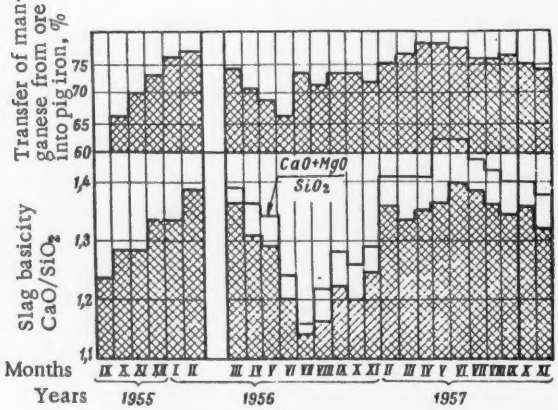


Fig. 3. The variation in the slag basicity (bottom diagram) and the transfer of manganese from ore into pig iron (upper diagram) during 1955-1957.

The content of manganous oxide in the slag depends on the slag basicity and pig iron temperature. When slag basicity increases, the content of manganous oxide in the slag falls markedly and hence, it follows that by increasing the basicity, one can substantially reduce the losses of manganese in the slag. The silicon content in pig iron exhibits similar effect.

The above findings were confirmed by a series of tests carried out on furnace No. 1. Beginning in March, 1957, the basicity was increased by means of the addition of MgO. Dolomite additions were very small owing to its shortage and poor quality.

In March, 1957, slag basicity was increased from 1.26 to 1.41, and, in April, 1957, while the basicity of the slag was kept the same, the silicon content in the slag was increased from 1.29% to 1.59%. In May, slag basicity was again increased to 1.47 by increasing the magnesium content in the slag (from 1.79% to 3.03%) but the silicon content in pig iron was reduced from 1.59% to 1.49%.

When the slag basicity and the silicon content in pig iron were increased, the transfer efficiency of manganese into pig iron rose, and the manganese losses in the slag, the consumption of manganese ore and the cost of pig iron decreased.

A subsequent reduction in the slag basicity (July-November, 1957) from 1.47 to 1.37 resulted in an increase in the manganese losses in the slag (from 10.9% in May to 15.1% in November) and in the fall in the transfer of manganese into pig iron (from 78% in May to 74% in November, 1957).

Calculations indicate that if ferromanganese is produced with basic slags and at an increased silicon content in pig iron, the annual saving on the operation of blast furnace No. 1 can constitute approximately 5 million rubles.

The new procedure of casting ferromanganese on the casting machine (less than 5 m/min velocity of the belts, the elimination of spraying the ferromanganese pigs on the machine belts and cooling the molds on the distance between the unloading end of the machine to the milk of lime sprayer) ensures a satisfactory particle size and strength of ferromanganese in transport and storage.

Editor's note: The measures, recommended in A. N. Redko's article, which contribute to a reduction in manganese losses can hardly be applied in the production of ferromanganese used for the deoxidation of rimmed steels, since they have a high silicon content.

A topic for discussion

RATIONAL METHOD OF CHARGING THE BLAST FURNACE*

L. Ia. Shparber

The Magnitogorsk Metallurgical Combine

At present, large blast furnaces are charged according to the scheme: bunker – scale car – skip. With the object of facilitating the duties of scale car operators, work is being carried out at blast furnace plants on the automation of the loading and weighing of charge materials, on the development of the delivery of the charge materials to the skip by conveyers and the transfer of materials from the bin to the skip by means of vibrating tube conveyers.

These methods, however, are not without serious disadvantages. The automation of the scale car is complicated and in that case the fine fraction of the sinter cannot be screened off. When conveyers are employed for charge delivery, a hot sinter cannot be used. Delivery by means of a vibrating conveyer necessitates a large number of separate feeders connecting separate bunkers with the tube.

The search for a rational scheme of charging the blast furnace led us to a solution which can be adopted either in full or in part, at plants under construction and during the reconstruction or scheduled repairs at operating plants.

In the complete scheme (see diagram) of rational charging, the hot sinter is unloaded into two bins (1) 850 cu m capacity each and thus it is possible to maintain sinter stock at 3000 tons. The bins differ from the existing ones for coke in that they are faced by rail blocks. Each bin is closed from below by two drum traps (cooled) with shutters. When the drum is rotated the sinter falls on to the screen (2), assembled from narrow gauge

* The method has been developed at the MMK (Magnitogorsk Metallurgical Combine) by engineers V. M. Zudin, N. S. Krivolapov, N. S. Reizov, I. I. Sagaidak, V.A. Shastin and L. Ia. Shparber.

rails from which the bottom plate has been cut off. Adjoining rails (bars) vibrate at different frequencies so that the openings between them do not get blocked. There are two screens on each side.

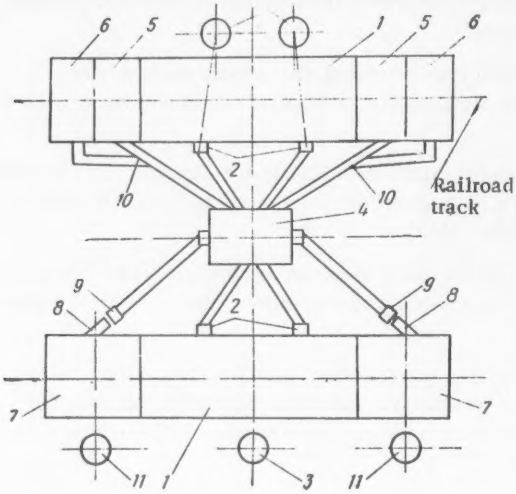


Diagram of the rational scheme for charging the blast furnace: 1) bins for sinter; 2) screens for sinter; 3) bins for sinter fines; 4) weighing equipment; 5) bins for the ore; 6) bins for additions materials; 7) coke screens; 9) weighing equipment for coke; 10) conveyers; 11) bins for coke fines.

The sinter fines are taken by a small skip elevator to the bins for sinter fines (3) and are used as returns at sintering plants.

The screened sinter enters the weighing equipment (4) closed from below by a door, which opens in the "ore charge" position when the sinter is transferred into the large skip.

When the required weight of sinter has been reached, the impulse for switching off the drum feeder and the vibrating screen is given by the weighing equipment; after the door has been closed the impulse for starting the feeder and the screen is given.

The ore and the additions are supplied from the bins (5 and 6) by means of light rubber conveyers to the weighing apparatus where the ore and the additions are weighed in turn. The weighing apparatus is interlinked with the charging units. The scheme becomes even more simple if the charge consists of sinter and coke only.

Coke is supplied from the bins (7), whose capacity is determined depending on the furnace volume. The bins have drum shutoffs through which coke is discharged onto the vibrating screens (8) and to the weighing apparatus (9) closed with sector traps. When the required weight of coke has been reached the drum and the screen are switched off and at the "coke charge" position the sector trap is opened. The coke falls through a short trough to a fast moving conveyer, is transferred to the main coke trough and hence into the skip.

The coke fines are collected by means of an ordinary coke-fines elevator or a pneumatic conveyer into the bins for coke fines (11).

The proposed method can be introduced in part during current repairs. In this case, the coke bin should be faced with blocks of railroad rails and used for the sinter. Ordinary screens have to be replaced by vibrating screens. The scale car can serve as the weighing apparatus.

Ore bins can be adopted for coke storage (three bins on each side of the former coke bin). The sinter fines from under the screens are collected by coke-fines elevators; two elevators for the removal of coke fines have to be installed. The ore and the additives are taken to the scale car by rubber conveyers.

The proposed scheme embodies the automation of the operations.

On a major overhaul, the cabin of the scale car operator ("B" post), situated along the inclined bridge, and two ore bunkers on both sides of the cabin, should be removed. In the space made avilable, the sinter bin which at the MMK can be 650 cu m capacity, i.e., 1000 tons of sinter, is installed. Then the equipment for the complete or a partial rational scheme, described above, for charging the blast furnace is set up.

The principal idea in the above scheme is the replacement of a large number of small bins by large capacity bins, on whose layout depends a full automation of the charging operation at a smaller stock house.

The rational scheme of charging the blast furnace can be adopted in any modern blast furnace plant. The adoption of this scheme makes it possible:

1) to sieve off the fine fraction of the sinter immediately before charging the sinter into the skip (it is also possible to separate the sinter into several fractions and charge the fractions separately into the furnace);

2) to introduce a full automation of charging operations;

3) to improve working conditions in the stockhouse and to eliminate the post of scale car operator;

4) to substantially reduce the dust content in the air in the stockhouse (the dust content in the air near the place of sinter discharge from the bin reaches up to 6000 mg/cu m when hot sinter is employed).

The compactness of the scheme allows for a reduction in the size of the blast-furnace plant. The introduction of the scheme does not require large capital expenditure.

IMPROVING THE METHOD OF PRODUCTION OF RESISTANCE ALLOYS IN ELECTRIC FURNACES

A. F. Kablukovskii, Ia. S. Leizerov and I. P. Solodikhin

The I. F. Tevosian "Electrostal" Works

Alloys of 1.09-1.40 $\Omega \cdot mm^2/m$ specific resistance at 20°C, on a nickel and iron basis, are used in the manufacture of heating elements for electric furnaces. To the first group of alloys belong the so called nichrome alloys of Kh20N80 and Kh15N60 types, and to the second group belong ferro-chrome-aluminum alloys of the OKh25Iu5 type. The chemical composition of the most commonly used alloys is given in Table 1 (in conformity with GOST 5632-51).

The whole production cycle of these alloys, from smelting to wire drawing, is distinguished by its labor consuming operations, and, until recently, the production involved a high percentage of rejected material, low output and a high manufacturing cost of accepted products. With the object of improving the quality of high-resistance alloys, an extensive program* on the improvement of the production technology of the nichrome alloy Kh20N80 and the chromal alloy OKh25Iu5 was carried out at the "Electrostal" Works.

TABLE 1

Chemical Composition of Electric Resistance Alloys, %

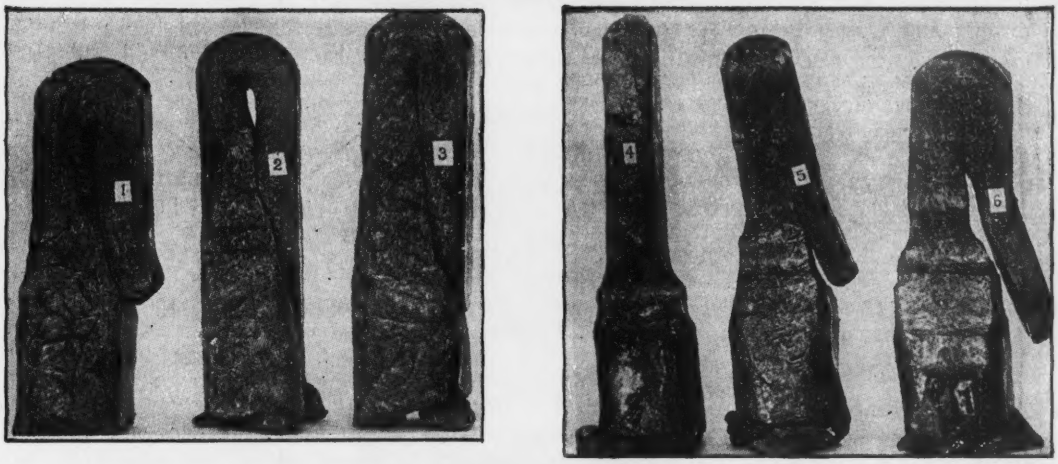
Steel grade	C	Si	Mn	Cr	Ni	Al	S	P	Fe
	not more than						not more than		
Kh20N80	0,15	0,50	1,50	20,0—23,0	75,0—78,0	—	0,025	0,030	Remainder
OKh25Iu5	0,06	0,60	0,70	23,0—27,0	Not more than 0,60	4,50—6,50	0,030	0,035	Remainder

Production of Nichrome Kh20N80

Kh20N80 alloy exhibits a high gas saturability which manifests itself either in a pronounced rise of the metal on solidification of the ingot in the mold, or in the form of deep seated voids or honeycomb blow holes. The presence of gas blisters very frequently caused the rejection of the ingots.

The Kh20N80 nichrome is produced in an electric arc furnace of 5 ton capacity provided with the basic lining of the walls and of the roof. In the old technique, the method of alloying electrolytic nickel with metallic chromium was employed. After the metals were melted, the slag was deoxidized with a mixture of calcium-silicon and coke. The final deoxidation of the metal was carried out either with lump calcium-silicon (5-6 kg/ton) containing 60-65% Si and 30-25% Ca, or with lump zirconium-silicon (4-5 kg/ton) containing 30-35% Si and 20-25% Zr. 5-10 minutes before the tapping 2-8 kg/ton of ferrotitanium was added to the molten alloy. The

*N. A. Shiriaev, V. E. Voinovskii, M. Ia. Dzugutov, V. S. Nikolskii, Iu. V. Vinogradov and others took part in the work.



Samples of the alloy after the malleability test.

alloy was top-poured into 500 kg circular ingots of 5% taper. Because of the low plasticity of the metal produced, 100-120 mm billets could not be rolled from the ingots. Therefore, the ingots were hammer-forged into the billets.'

As the basis for the new technological process of nichrome production, two main principles were accepted:

a) the Kh20N80 alloy should be made by means of remelting alloy scrap (wastes) containing nickel, chromium and titanium;

b) the pricipitation or submerged deoxidation of the bath with calcium-silicon or metallic calcium should be applied in addition to diffusional deoxidation through the slag.

By remelting highly alloyed, titanium containing scrap (chippings, shavings, gates and risers) it was possible in the early process of melting the charge to bind the nitrogen, dissolved in the liquid metal, with titanium into a stable chemical compound. Titanium nitrides float up and pass into the slag, their specific weight being 5.18 and their melting temperature 2930°C. Titanium combines not only with nitrogen but can also form hydrides with hydrogen partially dissolved in the metal. It was established that the ability of titanium to retain gases in stable compounds is well manifested when the titanium content in the charge constitutes 0.50-0.75%. Thus, on passing from the charge materials into the melt, titanium binds oxygen, nitrogen and hydrogen into compounds which do not decompose during the crystallization and solidification of the ingot.

In addition to diffusional deoxidation through the slag, the method of the deep deoxidation of metal by adding adding deoxidants in lumps — calcium-silicon or metallic calcium — was employed in the production of Kh20N80 alloy. The introduction of these strong deoxidants, by means of a rod into the melt, accelerated the process of removing from the alloy sulfur and oxygen compounds which impaired the plasticity of the alloy during hot deformation. Moreover, the introduction of excess calcium into the alloy protected it to a certain extent from being oxidized again during discharge from the furnace into the mold.

The charge is composed of the scrap of the alloy being produced (lumps, shavings, chippings), high nickel alloys of the Kh20N80T3 type, chrome-nickel-titanium alloys, etc. The total amount of the scrap in the charge constitutes up to 60%, and metallic chromium and electrolytic nickel constitute up to 40%.

Nickel grade N1 and metallic chromium, grades 1 and 2, are employed for alloying; limestone and fluorspar are screened and contain slight impurities of sulfur and silica.

The charge is melted at the full power input to the furnace. 25-30 minutes before the charge is completely melted, the deoxidizing mixture, consisting of limestone, fluorspar (if necessary), coke powder and calcium-silicon, is thrown onto the slag in the furnace. At the end of the melting period, the slag and metal is vigorously

stirred by means of rakers with nickel blades. After the whole charge has melted, the thickned slag is partly run off and the samples of metal are taken for spectral analysis, chemical analysis, a malleability and 180° bending test of 10-15 mm square section and for the determination of the temperature of the alloy. As a rule, cracks appear on the sides and edges of the first sample after the forging. The cracks open when the sample is bent through 180° (see Figure, sample 1). After the sample has been inspected, lump calcium-silicon (1 kg/ton) or metallic calcium (0.2 kg/ton) is introduced on a rod into the melt. A second sample, taken in 3-5 minutes, exhibits no cracks on the sides and edges after the malleability test (samples 2 and 3).

After a satisfactory plasticity of the alloy has been attained owing to the deep deoxidation, the diffusional deoxidation of the melt through the slag is continued. An additional amount of calcium-silicon (1 kg/ton) or metallic calcium (0.2-0.3 kg/ton) is introduced on a rod, about 15-10 minutes prior to the discharge of the alloy from the furnace. The samples taken after this addition exhibit seams and cracks (sample 4). In this case, the deterioration in the plasticity of the alloy is explained by the presence of an excess – above the permissible limit – of calcium, which has a low melting temperature and separates along the grain boundary on crystallization.

The presence of the theoretically required amount of calcium in the metal before it is tapped cancels the negative effect of the secondary oxidation of liquid alloy and contributes to the satisfactory plastic properties of cast samples and ingots on hot deformation (samples 5 and 6). The refining period lasts for 60 to 70 minutes. The temperature of the metal in the ladle after tapping is 1520-1540°C (measured with tungsten/molybdenum thermocouple).

The internal surfaces of the mold are thoroughly cleaned before filling but not lubricated. The time for bottom-pouring a 500 kg ingot up to the hot top is 30-60 sec, the time for top-pouring the ingot is 30-50 sec. After the mold has been filled, the hot top is covered with iron free thermite, containing 65% titanium dioxide and 35% aluminum powder, and with white slag. The ingots are cooled in the molds for at least two hours after filling. After being stripped and cooled, the ingots are transferred to chipping machines for the removal of surface defects.

In the production of the alloy by the new method, it was possible to replace the inefficient forging operation by rolling on mill 600. The machine-chipped round ingots, 450 kg in weight, are heated in holding furnaces to 1240°C in 8 hours. The ingots are then rolled on mill 600 into 125 mm square billets, according to the following rolling scheme: box passes (207.5 × 305 → 170 × 305 → 208.5 × 185 → 178 × 185 mm) – diamond pass (158 × 140 mm) – square pass 125 mm. The temperature of the billet at the end of rolling is not less than 900°C.

Some comparative data on the Kh20N80 alloy produced by the old and the new method are given in Table 2.

TABLE 2

Main Indices of the Production of the Kh20N80 Alloy by the Old and the New Methods

Method	Duration of the heat, hrs-min	Power consumption, kw-hr/ton	Material rejected owing to blowholes, honeycomb (ingot rise) and fractures, %	Loss of alloying elements, %		
				Cr	Ni	Ti
New	3-15	807	To 40	3.20	3.10	–
Old	2-55	646	To 3.0	2.70	3.00	62.6

With the new method it was possible to eliminate the defects due to blowholes and ingot rise, to improve the plasticity of the alloy on hot working considerably and to increase substantially the output of acceptable product.

The Production of the OKh25Iu5 Iron-Chrome-Aluminum Alloy

By the old method OKh25Iu5 alloy was made in 20-t electric ore furnaces; fresh charge, oxygen boil and alloying the metal with chromium and aluminum during the refining period by adding low carbon ferrochromium and primary lump aluminum, were employed. For the improvement of the malleability of ingots, 7 kg/t of alumino-calcium alloy, 0.7 kg/t of alumino-barium alloy and 0.6 kg/t of cerium were introduced into the melt before tapping. The alloy was bottom poured into 500 kg ingots.

The ingots were then transferred hot, in heated containers, to the forge shop. The temperature of the ingots at the time of charging into the holding furnace was not less than 700°C. Prior to being forged, the ingots were heated for 9 hours to 1200°C. The ingots were then additionally heated in the holding furnace and forged into 110 mm rounds. When the forging operation was completed, the temperature of the metal was not less than 800°C. The billets were cooled for 24 hrs in heated pits, then blisters, seams and others surface defects were thoroughly removed by chipping and the billets were rolled on the mill 300.

With the object of reducing losses due to the amount of rejected product, increasing the output of useful product and reducing the production cost, the "Electrostal" Works developed a new method of manufacturing the OKh25Iu5 alloy, the following principles being accepted as the basis for the new method:

a) the alloy to be made in 5-t electric arc furnaces; a fresh charge, the ore boil and the addition of chrome-aluminum alloy during the refining period being employed;

b) a preliminary deoxidation of the bath with lump aluminum to be applied and metallic titanium to be used;

c) carbon tetrachloride to be employed for the protection of liquid metal from oxidation on pouring.

The most typical heats of the new method are carried out as follows.

The charge is made up to 1500 kg carbon steel scrap and 1620 kg of steel-45 scrap. The furnace is charged from above, with a total of 3120 kg of materials. The charge is melted at the full power of the furnace (1350 kw). 65 minutes after the complete melting of the charge, a sample for chemical analysis is taken. The metal contains 0.45-0.55% C, 0.18-0.28% Mn; 0.015-0.025% P and 0.15-0.25% Ni. The decarbonization and the dephosphorization of the bath is carried out with iron ore or oxygen.

During the boil period, the slag is run off and renewed 3 or 4 times. The manganese content is not controlled during the oxidation period. The oxidation period lasts for 60-80 minutes. The boiling of the bath is completed at 0.03-0.02% carbon content. After the oxidation slag is run off, the liquid steel is deoxidized by the addition of 15 kg of lump aluminum. 50-60 kg of lime and 15-25 kg of fluorspar is thrown onto the clear surface of the metal. The oxidation slag is fully removed in order to prevent a high content of silicon in the finished alloy.

The chrome-aluminum alloying addition is heated to 700-850°C in special trays and is added to the melt through the roof of the furnace in one or two batches. The total amount of the chrome-aluminum alloying addition is 1600-1650 kg (chemical composition: 0.03-0.04% C, 0.55-0.65% Si, 63.0-65.0% Cr, 17.0-18.5% Al; up to 0.006% S, 0.006% P; Cu – 8 grains; Pb, Sn, As, Zn, Bi – not more than one grain each). Any deficiency in chromium is made up by the introduction of ferrochromium, grade 00000.

After the alloying addition has been melted, a sample for a complete chemical analysis is taken. 5 minutes prior to tapping, 10 kg of metallic titanium, or a corresponding amount of titanium scrap, is added to the bath. Expensive and scarce ferro-aluminum-zirconium, aluminum-barium and aluminum-calcium alloying additions are not employed. The temperature of the alloy in the ladle is measured by a tungsten/molybdenum thermocouple. The alloy is tapped at 1580-1620° and is bottom poured into a 500 kg ingot. 50-100 cu c of carbon tetrachloride is poured into each mold before it is filled. The time of filling the mold with liquid metal is 30-40 sec. 40-50 minutes after the last set of molds has been filled, the ingots are removed from the molds and placed in heated-up thermostatic containers which are taken by truck to the forge shop.

It is seen from Table 3 that the main cause of the rejection of the OKh25Iu5 alloy ingots and billets were defects due to cracks and fractures which were markedly reduced after the new method of production was adopted.

The OKh25Iu5 alloy made by the new method is distinguished by a low silicon content and smaller deviations in the aluminum and chromium content. In spite of the fact that the chromium content in the alloy made by the new method is higher (25.0-26.0%), the defects due to blisters, cracks and fractures are considerably reduced. This fact indicates a substantial improvement in the plastic properties of the alloy during hot deformation. Warm rolling, cold drawing and broaching from 8 mm diameter downwards, of the metal from those heats also gave satisfactory results.

An extensive adoption of the new method of producing cheaper and strong chromium-aluminum alloys will present the ferroalloy industry with the demand for supplies of chrome-aluminum alloying material of not

TABLE 3

Losses Due to Defective OKh25Iu5 Alloy Products, Rejected Because of Defects Resulting from the Smelting and Refining Process

Method of manufacture	Years	Defective products, %			Cost of production of 1 ton of crude ingots, rubles
		Total	Out of total		
			losses in runners and due faulty chemical composition	blisters, cracks and fractures	
	1955	31.0	0.6	99.4	–
Old	1956	24.7	–	100	–
	1957	46.1	40.3	59.7	5373.83
New	1957	2.8	–	100	4631.47

more than 0.03% carbon and not more than 0.40% silicon content.

As a result of the introduction of the new method of manufacturing electric resistance alloys based on nickel and iron, it became possible to eliminate defects in nichrome production due to ingot rise and blow holes as well as fractures on hot deformation; to substantially reduce the defects due to blisters, cracks and fractures on hot deformation of the OKh25Iu5 alloy; to increase the output of useful products; to reduce the cost of the alloy and to improve the technical and economical indices of furnace operation.

THE PRODUCTION OF TRANSFORMER STEEL UNDER VACUUM

I. S. Prianishnikov

The I. F. Tevosian "Electrostal" Works

In 1954, at the steelmaking plant of our Works, experimental heats of transformer steel were carried out under vacuum (0.5-1.0 mm Hg pressure) in the experimental furnace of 150 kg capacity. On the basis of 16 experimental heats, the following principles of the technology of production were established:

1. Vacuum should be applied only after the charge has melted. Otherwise, during the melting period the metal boils, bridges are formed and their elimination necessitates the return to normal pressure, and the upper part of the furnace lining is frequently damaged.

2. It is preferable to pour the metal without vacuum or in an inert gas atmosphere because it is difficult to control the flow of metal under vacuum in the OKB-215A furnace.

3. Ferrosilicon should be added under vacuum, a few minutes before tapping. Before silicon is added, the metal should be deoxidized under vacuum. When this condition is observed, much less slag is formed and the metal is not oxidized when the vacuum is removed.

4. In order to prevent the volatilization of silicon from the melt, the liquid metal with silicon added to it should be kept under 0.5-1.0 mm Hg pressure for not more than 10 minutes. On a longer exposure, the efficiency of the pumps deteriorates markedly and the evacuation duct rapidly gets clogged. An intensive volatilization of silicon under vacuum is observed if the metal is not covered with slag. If the metal with silicon added to it, is kept under a layer of slag, no volatilization of silicon takes place.

The result of the determination of the gas content,* nonmetallic inclusions and magnetic properties of the metal showed that the metal treated under vacuum contains considerably less gases and nonmetallic inclusions

*The gas content was determined by the method of vacuum melting, on samples taken after the ingots were forged into billets.

than ordinary metal (see Table). The nonmetallic inclusions are larger and of globular shape.

Even if the metal treated under vacuum has a high silicon content (4.12%), its plasticity, however, is higher and its magnetic properties are better than in the case of metal from ordinary heats.

Properties of the Transformer Steel

Property	Heat	
	open	under vacuum
Content:		
oxygen, 10^{-4}%	145-195	19-26
hydrogen, 10^{-4}%	2-4	0.5
Content of nonmetallic inclusion, 10^{-4}%	340-500	40-70
Out of these inclusions, % of total content:		
Al_2O_3	62-80	–
SiO_2	3.5-24	70-95
Wattage losses, w/kg:		
steel R10	0.90-1.18	0.74-0.85
steel R15	2.05-2.70	1.67-1.78
Magnetic permeability, g • sec/oersted:		
initial	500-640	1100-4000
maximum	5400-9000	10,400-13,900
Coercive intensity, oersted	0.352-0.535	0.220-0.283

Two methods of producing transformer steel under vacuum were developed in 1955.

The first method. The charge is composed of Armco type, low carbon iron (the "Serp i Molot" Works) or of Sulin sponge iron, remelted beforehand in the arc furnace. 75% ferrosilicon is added in order to attain the required content of silicon. Vacuum is applied when all the charge is in the furnace and has been partly melted, i.e., when the formation of bridges cannot take place any more. The vacuum is applied gradually, so as to avoid violent boiling and splashing of the metal onto the upper part of furnace lining. The evacuation of gases is controlled by a valve, positioned between the furnace and the pumps.

The fully melted metal is kept under vacuum until it ceases boiling (20-30 min). In all, the metal is kept under vacuum for 80-95 min. This time of exposure was chosen on the basis of a study, carried out earlier, which showed that during the period of heating and melting the metal charge under vacuum, up to 70-80% of the gases contained in the metal can separate out.

Ferrosilicon is added under vacuum through a dosing apparatus. The melt with ferrosilicon added, is kept under vacuum for 8-10 min. During the dissolution of ferrosilicon, when its concentration on the surface of the melt is high, silicon volatilizes rapidly and for this reason the vacuum pumps are switched off.

If a large quantity of slag forms on the surface of the melt, the vacuum is removed, the slag is quickly discharged and the metal is tapped. If there is no slag, the metal is tapped under vacuum. Six heats were carried out according to the above method. The metal was satisfactory in chemical composition and no defects appeared in the course of subsequent processing. The life of the furnace (crucible) was 8-10 heats.

The second method was designed with a view to producing a better quality metal of lower sulfur and phosphorus contents. The metal was melted at atmospheric pressure under the layer of the slag of an ordinary composition (15% MgO, 15% CaF_2, 70% CaO); carbon-electrode chips in the amount within 0.05% carbon content in the metal, and roasted iron ore (0.1% by weight of the charge) were added to the charge.

During the melting period the metal was boiling. After the charge had melted, the slag was removed and new slag was formed with lime and fluorspar. After the metal had been kept under this slag for 7-10 minutes, the slag was discharged and the diffusional deoxidation of the metal with the slag made up of 3 kg lime and 1 kg aluminum-lime mixture, was carried out. The metal was kept under this slag for 10 minutes, the slag was then completely removed and the deoxidized metal was kept under vacuum for 20 minutes. No vigorous boiling was observed when the above procedure of keeping the metal under vacuum was adopted.

After ferrosilicon was added, the metal was kept under vacuum for 15 minutes. The procedure of pouring was the same as in the first method. 30 heats were carried out by this method; out of these, 10 were rejected at the plant owing to incorrect chemical composition before the metal was passed to other plants for processing. The content of sulfur and phosphorus was lower in the metal produced by the second method than in that produced by the first method.

The life of the furnace (crucible) was, however, half as long for these heats as for the heats carried out by the first method.

Actual silicon content in the heats of the same theoretical silicon content exhibited pronounced fluctuations and constituted from 3.3 to 4.2%.

The duration of the heats increased by 30 minutes on the average. Power consumption increased by 500-600 kw-hr per ton. The following conclusions can be drawn from the above work:

1. Transformer steel and alloys of similar type can be economically produced under 1 mm Hg pressure in evacuated induction furnaces of not less than 1 ton capacity.

2. It is uneconomic to remove sulfur and phosphorus directly from the vacuum furnace. The initial materials should be free of sulfur and phosphorus.

3. It is uneconomic to keep already deoxidized metal under vacuum.

The main principles of the technology of producing transformer steel in a vacuum furnace are:

1) The application of the vacuum begins at the end of the melting period and is completed when metal boiling subsides.

2) The metal is refined under vacuum with carbon; after metal boiling subsides, carbon-electrode chips are introduced into the bath through the dosing apparatus; an intensive boil results and continues for 10-30 minutes depending on the amount of carbon added.

3) Ferrosilicon is added under vacuum, and the metal containing silicon is kept under vacuum for not more than 10 minutes.

4) The metal should be poured under vacuum or in an inert atmosphere.

OPERATIONAL EXPERIENCE ON EXPANSION CIRCUIT BREAKERS AT ELECTRIC FURNACES

I. T. Danilin

The "Krasnyi Oktiabr" Works

The 10 ton capacity electric furnace with transformers of 5000 kva, 6 kv and 480 amp nominal current were equipped with VMB-10, 600 amp, oil circuit breakers. In the course of operation, serious defects in these switches were encountered, the chief of these defects being a rapid wear of contacts and the deterioration of the transformer oil.

Initially, the circuit breakers were repaired only during the cold repairs of the furnaces, i.e., every 12-15 days; later on, the period between two consecutive repairs of the circuit breakers was reduced to 5-6 days. But even in such a short time of operation the dielectric strength of the transformer oil used to fall from 50 to 16-18 kv. The furnaces stood idle for 1-1.5 hours during the repairs of the circuit breakers,the output of the furnaces being thus affected.

The main disadvantage of the VMB-10 circuit breakers is the danger of explosion. A breakdown which involved the bursting of the oil tank of the circuit breakers, took place at one of the electric furnaces at the end of 1956. The cause of the breakdown was the wear of the contacts.

Cutting down the service period of the circuit breakers to 5 days did not eliminate the breakdowns completely.

Frequent repairs caused long idling periods of the furnaces, an excessive consumption of transformer oil and considerable labor expenditure.

In spite of the fact that the circuit breakers of VMB-10 type are obviously unsuitable, our electric industry continues to produce only these circuit breakers because no better switches have been developed in the last 10 years. Neither are circuit breakers of the VMG-133 type found to be suitable for electric furnaces because of their small capacity and poor mechanical strength.

The employees of the "Krasnyi Oktiabr" Works replaced the VMB-10 switches with the Siemens expansion switches for 20 kv, 600 amp and of 500 mva breaking power.

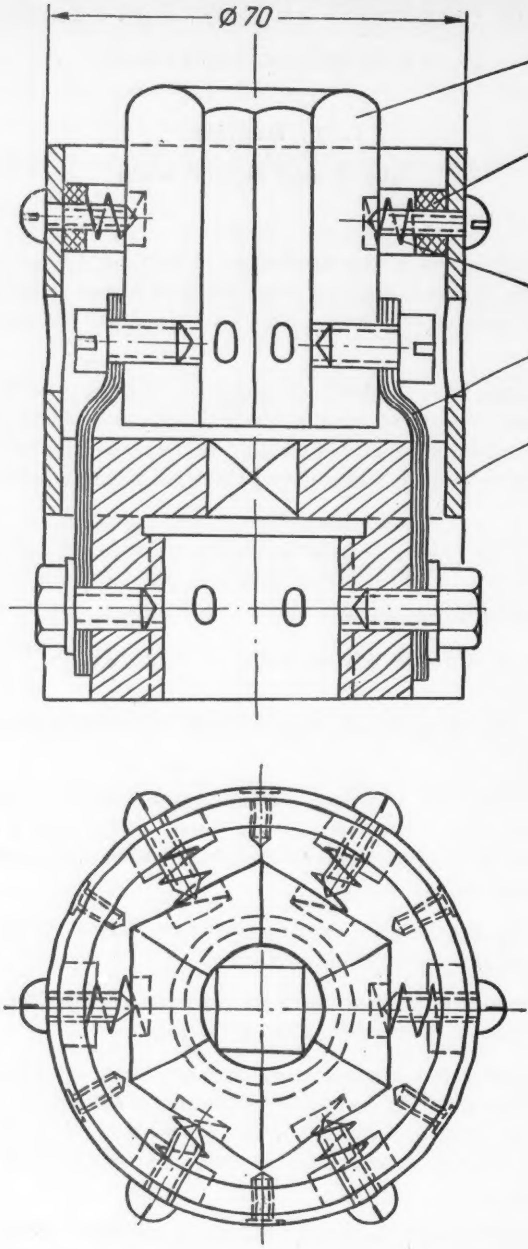
Compressed air at 5 atm pressure was taken from the general air main of the plant. A specially installed compressor can be switched in should the pressure in the mains fall.

When these circuit breakers were introduced a new difficulty arose: the contacts of rosette copper were found so weak that they would not last even for one campaign of the furnace, so that it was necessary either to build them up with metal or to replace them; the production, however, of rosette copper is very difficult and labor consuming. Therefore, the original contacts were replaced by new contacts, made on the pattern of the VMG-133 contacts (see Figure).

The expansion circuit breakers were found to be reliable in operation. In six months of operation not a single idle period caused by a breakdown was due to the circuit breakers. Now the circuit breakers are checked only during the cold repairs of the furnace. The contacts hardly get worn down at all in the period between the repairs; in the course of the check up they are only cleaned. One can be quite confident that the contacts will remain in service for more than a year without replacement.

The adoption of the expansion circuit breakers, which are filled with distilled water, resulted in the elimination of unnecessary stoppages of the furnaces and excessive consumption of transformer oil.

In conclusion one would wish that our electric industry would develop and start the production of electric furnace circuit breakers making use of the experience of Soviet works.



The contact rose of the expansion circuit breaker:
1) contacts; 2) nonmagnetic metal ring; 3) spring; 4) thin copper foil; 5) brass sleeve.

PNEUMATIC DRIVE FOR RETURNING THE MIXER ON LOAD TAKE-OFF

V. S. Brazhnik

Deputy Head of the Design Department at the "Krivorozhstal" Works

Mixers made according to standard designs are equipped with rack and pinion rotating gear consisting of toothed rail connected through a hinged joint to the stirrup of the mixer and actuated by an electric motor through reduction gears and two open gears. One of the virtues of this mechanism is that when the power is switched off, the mixer returns to its initial position. If the center of gravity of the mixer is displaced, owing to a deviation in the dimensions of the lining from the design values, the deformation of the jacket or some inaccuracies in the mountings of the mixer, it sometimes happens that the mixer does not return by itself into the initial position. Under those circumstances, serious breakdowns, involving heavy damages, are unavoidable when the power is switched off or the motors stall at the time of the discharge of molten iron.

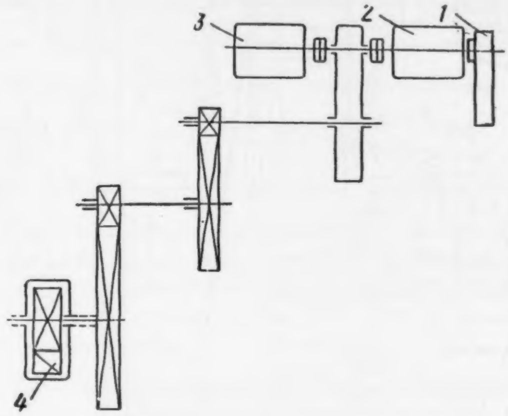


Fig. 1. Diagram of the layout for the installation of the pneumatic drive for returning the mixer: 1) pneumatic drive; 2) electric motor No. 2; 3) electric motor No. 3; 4) rack.

The 1300-t mixer of the "Krivorozhstal" Works has been equipped with a special pneumatic drive for returning the mixer on power switch-off. The device is attached through a closed coupling to the second shaft of one of the electric motors (Fig. 1).

Data on the drive are given below:

Diameter of the pneumatic cylinder, mm	150
Piston stroke of the pneumatic cylinder, mm	110
Working pressure, atm	4-6
Torque on the shaft of the motor, kg m	$M_{kp} = 185$
Diameter of the slide valve, mm	32
Stroke of the slide valve, mm	12
Total weight of the apparatus, kg	670

The drive consists of a frame, ratchet gearing, pneumatic cylinder and slide valve (Fig. 2).

In its normal position, the ratchet (3) is disengaged and, owing to a special construction of the ratchet gearing, the ratchet wheel (2) rotates freely together with the shaft of the electric motor.

In the idle position, the lever (4) is maintained in horizontal position by its own weight and by the holding spring (5). Under the action of the piston and the weight of the stem, the ratchet is turned on the axis O_2 and is

disengaged from the wheel. The maximum clearance between the ratchet and the wheel is 3-4 mm.

When the mixer has to be returned into the initial position by means of the pneumatic drive, the operator of the mixer sets the three-way air valve at starting position. When the piston moves upwards, the ratchet turns about the axis O_2 up to the stop B, engages with the ratchet wheel and, executing a rotary motion together with the lever (4) about the axis O_2, turns the ratchet wheel by one notch.

At the moment when the piston reaches its extreme top position, the lower stay of the cross beam hits the arm of the crank which actuates the valve. The ratio of the two arms of the crank is selected in such a way that the 1.5 mm displacement of the cross beam at the extreme positions of the piston corresponds to the full 12 mm displacement of the valve.

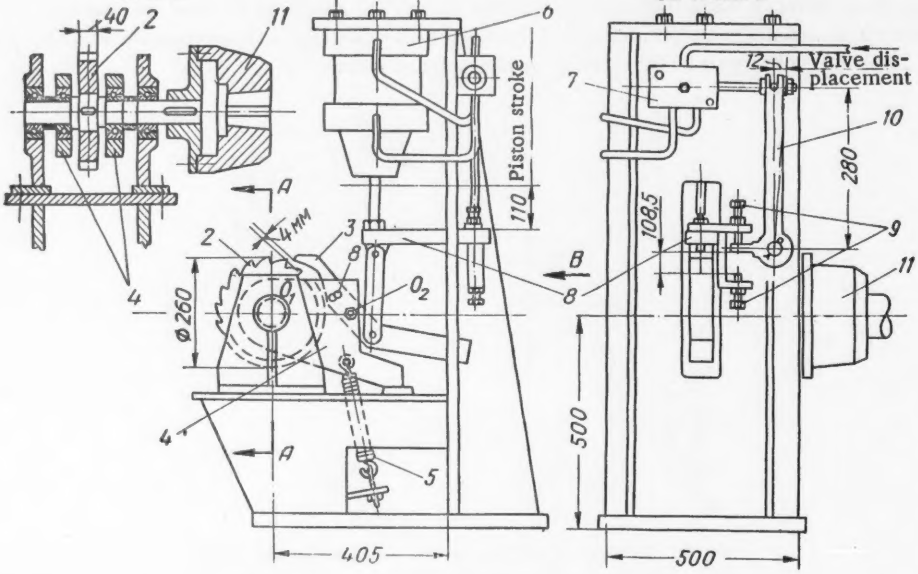


Fig. 2. Pneumatic drive for returning the mixer on power take-off:
1) frame; 2, 3, 4 and 5) ratchet wheel device; 6) pneumatic cylinder; 7) valve; 8 and 9) cross beams; 10) the crank of the valve; 11) disc coupling.

It was found from experience that it takes several seconds to return the mixer into the original position.

When the three-way valve in the air main is opened to atmosphere, all the levers of the ratchet wheel device return to their initial position under their own weight and under the action of the holding spring, and the ratchet wheel is automatically disengaged from the mixer's main drive.

The installation of the pneumatic drive ensures safety in the operation of the mixer in case of the stoppage of the main drive; no large expenditure is required for the installation and this device can be recommended for other works.

POSSIBLE OVERLOADING OF ELECTRIC ARC STEELMAKING FURNACES

Engineers A. I. Sapko and Z. I. Sapko

In "Metallurgist" No. 1, 1958, an article on "Possible Overloading of Electric Arc Steelmaking Furnaces" by Cand. Tech. Sci. E. N. Astrov was published as a topic for discussion. On the basis of the analysis of the operation of 20 to 40 ton electric arc furnaces at the "Dneprospetsstal" Works, we are considering, in the present communication, the problem of raising the output of the furnaces by increasing the weight of the charge, i.e., by overloading the furnaces above their design capacity.

First of all, let us define what is understood by the design capacity and the overloading of the furnace. As is known, the dimensions of the bath of a furnace are determined on the basis of the main determining parameter viz. the ratio of the diameter of the surface area of the metal, D_{met}, to the depth of the bath, H_{met}. Previously, on the majority of furnaces at home and abroad, the ratio $\frac{D_{met}}{H_{met}} = 6.6$ was accepted, since it was considered that, from the point of view of metallurgy, a "shallow" bath is preferable in the production of quality steels.

The All-Union Conference of Steel Workers has very strongly criticized the design organizations which designed large furnaces based on the ratio $\frac{D_{met}}{H_{met}} = 6.6$ and, on the basis of the analysis of the operation of 30 to 40 ton electric furnaces, has fixed the optimum value of the ratio $\frac{D_{met}}{H_{met}} = 5.35$.

If in the design of an electric furnace, a $\frac{D_{met}}{H_{met}}$ ratio higher than 5.35 is chosen, the geometrical parameters of the furnace are increased in advance and the design capacity of the furnace projected is too low. In the course of operation, the "undercharging" of the furnace becomes apparent and the operators begin to "overload" the furnace (or more accurately, to operate the furnace at the rated design capacity). But, it is not always possible to utilize the revealed reserve for the output increase because of the disparity between the increased weight of the charge and the lifting capacity of pouring crane.

The fact, pointed out above, that the design organizations were misled in the choice of a lifting capacity of the cranes by the excessive value of the $\frac{D_{met}}{H_{met}}$ ratio equal to 6.6 for 30 and 40 ton electric furnaces, was confirmed in the course of operation of one of the large plants at the "Dneprospetsstal" Works. At that plant, the rated overloading of the 40 ton electric furnaces by 30% resulted in the 100/25/5 ton crane being operated at the maximum lifting capacity; to replace that crane by a 125 ton crane would necessitate strengthening or replacing the supporting girders.

In the past 5 years, all furnaces at the Works have been modified to operate at an increased output. Several methods of modification, with and without an increase in the transformer capacity, were tried out.

On comparing various methods, the personnel of the "Dneprospetsstal" Works came to the conclusion that it is undesirable to overload the furnaces* by more than 30% without an appropriate modification of the furnaces and an increase in transformer size (this would be practically equivalent to the installation of a larger capacity furnace).

The following conclusions can be drawn on the basis of the operational experience of the "Dneprospetsstal" Works :

1. to increase the output of electric furnaces by means of overloading them by more than 30% compared with their design capacity (at $\frac{D_{met}}{H_{met}} = 6.6$) is considered is uneconomic owing to an overall decline in the economic and technical indices of the furnace;

* $\frac{D_{met}}{H_{met}}$ ratio for these furnaces is 6.6.

2) the construction of the furnaces with a high $\frac{D_{met}}{H_{met}}$ ratio (above 5.35) is misleading for the design organizations when they choose the load-lifting equipment, the result being that the furnace unit is not fully utilized (due to the disparity between the weight of the heat and the load-lifting capacity of the crane);

3) the output of the electric steelmaking furnace should not be increased by overloading it but by:

a) improving operating conditions in the electricity supply (increase in the secondary voltage of the transformer, improvement of the self-regulating systems for electric arc furnaces, development of better compensated short networks, etc);

b) intensifying the steelmaking process by the application of oxygen;

c) improving the design of the mechanical and electrical equipment of the electric arc furnaces.

THE ROLLED AND TUBULAR PRODUCT INDUSTRY

LOOPING GUIDES ON CONTINUOUS MILLS

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Roll-design engineer

The Magnitogorsk Metallurgical Combine

In the operation of continuous mills the rule of constant volume of metal passing through each stand of the mill per unit time is observed.

For a number of successive stands, employed in the rolling operation, this rule can be expressed as:

$$F_1v_1 = F_2v_2 = F_3v_3 = \ldots = F_nv_n = \text{const},$$

where F_1, F_2, etc. – the cross sectional area of the piece leaving the stand;
v_1, v_2, etc. – the speed at which the piece leaves the stand.

In practice, however, it is very difficult to adhere to the above condition since the cross sectional area of the piece changes continually owing to the variation in the coefficient of friction, changes in the temperature of the rolled piece, the wear of the rolls, etc.

On the other hand, if each stand is driven separately, the rolls can rotate at different speeds and hence the rolled piece is either stretched or forms loops between the stands.

To obtain a section of good quality and correct cross sectional area, it is desirable to avoid keeping the rolled piece under tension between the stands. The tension in the piece results in the formation of three zones on the rolled piece – front, middle and end – which provide a measure of the extent to which the grooves of the given stand are filled with metal and which therefore have different cross sectional areas. The cross section areas of the front and the end zones are larger than the cross section area of the middle zone of the rolled piece. The greater the tension, the greater is the difference in the cross section area. All this makes the adjustment of the mill very difficult. Most frequently, the foreman adjusts the mill according to the cross section area of the middle zone of the rolled piece. In many cases, however, this manner of mill adjustment results in extensive laps on the finished section (Fig. 1).

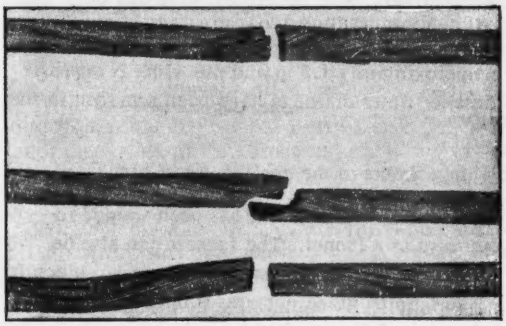


Fig. 1. Laps on finished section.

Because of the excessive cross section of the end part of the piece, the grooves are "overfilled" and the fin formation takes place either on one side or on both sides of the piece, the fins resulting in the appearance of laps on the final product.

At present, no reliable methods are yet available for the determination of the extent of stretching the piece between the stands. It is better, therefore, in the process of rolling to allow the formation of a small loop between the stands or between separate groups of stands instead of tension in the piece. This is most desirable in the case of rolling pieces of small cross section area (light-section and wire-rod mills), since if such sections are stretched between the stands, a

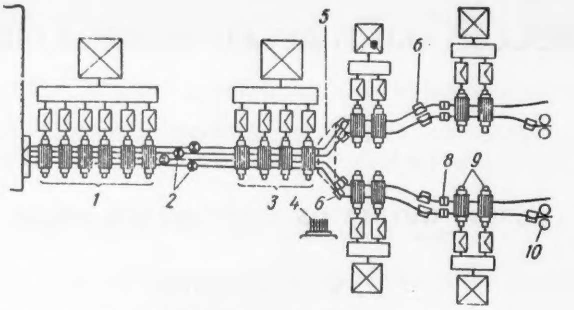


Fig. 2. The layout of the wire-rod mill 250:
1) roughing group; 2) rotary shears; 3) first middle group; 4) coilers for rejects; 5) pit for loop formation; 6) looping guides; 7) second middle group of stands; 8) emergency shears; 9) third middle group of stands; 10) finishing stands with vertical rolls.

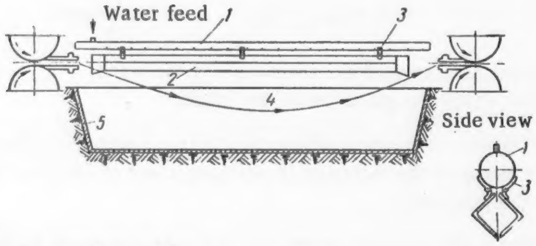


Fig. 3. Looping guide:
1) pipe for cooling the angle bars of the looping guide; 2) looping device constructed of angle iron; 3) hinge joints between the cooling-water tube and the angle bars; 4) loop; 5) pit for the loop.

relatively large deformation of the cross section area of the intermediate product takes place with serious consequences for the final section.

At the continuous wire-rod mill 250 (Fig. 2) of the Magnitogorsk Metallurgical Combine, looping guides of an original design (Fig. 3) have been installed behind the 10th stand, and at the light-section mill 250 – behind the 6th stand.

In the space between the groups of stands where, from the technological point of view, it is most important that no stretching takes place, pits, lined with iron plate (or, alternatively, with concrete), were dug out. The dimensions of the pit depend on the size of the loop.

At the mills 250 of our Combine, the depth of the pits is approximately 1.5 m and the width is approximately equal to the length of the roll barrel of the stand. Outlets for water drainage have been provided in the bottom of the pit.

The looping guide consists of two angle bars attached by hinge joints to the cooling-water pipe.

For a better entry of the rolled piece into the looping guide, one half of a funnel has been welded to each angle bar; hence, the front end of the looping guide has the form of a funnel. The funnels can also be welded to the angle bar at the delivery end. The opening of the angles and the discharge of the rolled piece in the initial moment of loop formation will thus be facilitated. The looping guide operates simply and reliably; under a small pressure of the rolled piece the angles open and the piece falls out, forming a loop.

The adoption of the looping guide of the new design assists in raising the quality of production and in improving the technical and economic indices of the rolling process.

OPERATION OF HOLDING FURNACES WITH INJECTION BURNERS AND LOW-PRESSURE BURNERS

D. E. Krasnozhen

The Stalino Metallurgical Works

The working space of a heating holding furnace is divided into zones, each of which has distinctive temperature and thermal conditions. The first zone serves for fuel combustion, and the second – for the utilization of the heat of the combustion products from the heating zone.

For given limiting temperatures of the gases and those of the metal, the intensity of heat exchange in the furnace rises in relation to the length of the heating zone.

In practice, some heat is transferred from the first zone to the second on account of the incomplete combustion of the fuel in the heating zone and its final combustion in the holding zone, and also through direct heat radiation from the first zone to the second.

The direct radiation from, and the incomplete combustion in the heating zone, lower the heat transfer rate in that zone and, consequently, the affect temperature and the heating regimes in the furnace. The necessary factors to ensure optimum operating conditions are:

1) complete combustion of the fuel within the heating zone;

2) when fuel is supplied to the heating zone, the temperature of the gases along the furnace must not fall,

3) direct heat radiation from the heating zone to the soaking zone must be at its minimum.

These conditions can be fulfilled if the burners are rationally designed and appropriately positioned in the furnace, and the optimum dimensions of the working space of the furnace are established.

To increase the rate of heat exchange in a three-zone holding furnace, the temperature of the gases in the first zone is increased. Therefore, there is a high temperature gradient in the cross section of the material leaving that zone. To equalize the temperature, the metal is kept in the soaking zone. Experience has shown that the three-zone method of heating metal is more appropriate than the two-zone method.

Operation of the Furnace with Low-Pressure Burners

The furnace of the 250 mill – three-zone, two-row and providing for two-side heating of metal – was designed to operate with recuperators and low-pressure burners of concentric type, using mixed gas of 1800-2000 kcal/cu m calorific value at a pressure of 50 mm of water. The burners were arranged in three groups – one in the front end of the soaking zone, another in the upper end and the third in the lower end of the heating zone. There were 12 burners in all, four in each group. The distribution of the heating capacity of the burners was as follows: the upper end of the heating zone – 40% of the total heating capacity of the furnace, the lower end of that zone – 40%, and the front end of the soaking zone – 20%. Owing to design defects and poor workmanship the recuperator was soon out of order. The burners were using cold air and, consequently, the calorific value of the gas had to be increased to 2700 – 3000 kcal/cu m.

The burners, designed by the Stal'proekt, were gas nozzles inserted into the air ducts of the furnace. Their main defect was low exit velocities of gas and air. At the maximum input per burner in the upper zone of 375 cu m/hr (of gas of 2700 kcal/cu m calorific value), and 450 cu m/hr in the lower zone, the exit velocity of the mixture in the nozzle was rather low – 13.2 m/sec and 16 m/sec for the upper and the lower zones respectively.

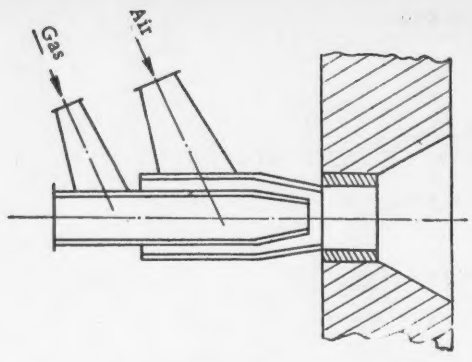


Fig. 1. The low-pressure burner of Stal'proekt design after modification.

Furthermore, entering the air duct of large cross section and very low air velocity the gas stream mixed inefficiently with the air and, hence, after leaving the mixer, the air and the gas separated into layers, the air (as the heavier component) spreading over the surface of the metal and increasing its oxidation. The flame tongue was elongated and therefore the combustion was not completed in the heating zone but continued in the holding zone.

Owing to the inefficient mixing of gas and air, much heat (up to 7%) was lost because of incomplete combustion of the fuel.

In 1954, specific fuel consumption for the furnace was 81 kg – 107.5 kg per ton of accepted product. The productive capacity of the working hearth area was 250-450 kg/sq m/hr.

The burners were designed for 17 m/sec exit velocity of the gas mixture. It was not possible to attain a higher gas velocity at the exit from the gas nozzle owing to inadequate gas pressure before the furnace in the common gas manifold. New burners with a reduced cross section of the gas nozzle (Fig. 1) – 90 mm diameter nozzles for the upper part of the furnace and 95 mm diameter nozzles for the lower part to replace the old 100 mm diameter nozzles – were designed and installed during the cold repairs in January 1955.

The new burner consists of two concentric tubings having a tangential air feed to provide turbulence.

The furnace efficiency improved considerably after the new burners had been installed in the upper and the lower parts of the heating zone of the furnace.

As a result of the increase in the gas and air exit velocities and in the turbulence of the stream, the mixing of the gas with the air improved markedly, the flame produced by the burner became shorter, the combustion of fuel was completed within the heating zone where the working temperature increased and, consequently, the output of the furnace increased.

The productive capacity of the working hearth area increased to 400 kg/sq m/hr. The consumption of conventional fuel decreased markedly (from 95 kg per ton of accepted product to 78-76 kg per ton).

The Operation of the Furnace with the High-Pressure Burners

The three-zone, two-row, recuperative holding furnaces, No. 1 and No. 2, fired from two sides, were built according to a design by the Stal'proekt. The working hearth area of the furnace is 75.5 sq m.

The furnaces are equipped with high-pressure (1500-1800 mm of water) injection burners (designed by the Stal'proekt, Fig. 2) each of 100 cu m/hr capacity. The air for the burners is injected through the air ducts of the recuperator and is a supplied to the burners through an insulated air main. In all, the two furnaces have 16 burners including 4 in the soaking zone and 12 in the heating zone, of the latter number 4 are side burners (the upper part of the zone) and 14 are bottom burners. The internal diameter of the gas nozzle of the burners is 60 mm.

The fuel is mixed (coke oven and blast furnace) gas. The ceramic recuperator has 108 cu m volume.

The temperature of the combustion products before the recuperator is 900°C, after the recuperator it is 460°C. The air is heated to 450-550°C.

The furnaces are employed for heating carbon steel and low alloy steel. The design output of the furnace is 35 tons of ingots or 45 tons of slabs per hour. The furnaces operate on a dry hearth; ingots and slabs travel on four water cooled sliding tubes which are supported by 15 transverse tubes.

In the course of operation, it was found that the burners require a minimum excess air (1.02-1.05), provide for an efficient mixing of gases and produce almost flameless combustion. At present, the calorific value of the

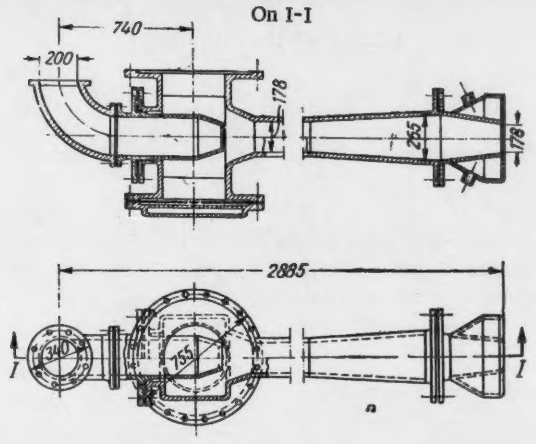


Fig. 2. High pressure burner.

gas is below the rated value and constitutes 1230-1300 kcal/cu m, since the air is heated to 450-500°C in the recuperator and, consequently, there is no need for burning gas of 1600 kcal/cu m calorific value in the furnace as provided for in the design. The required temperature in the furnace is attained when gas of 1300 kcal/cu m calorific value is used.

The amount of combustion products increases by only 3% and this has practically no effect on the draft.

With the existing cross section of the gas nozzles, the use of such a gas in the injection burners of the upper soaking zone is more advantageous because when the calorific value of the gas mixture is increased to more than 1300 kcal/cu m, the combustion of fuel becomes incomplete.

Investigations and the operation of the furnaces showed that, on designing them, the Stal'proekt made an error in the distribution of the heating capacity of the burners in the zones of the furnace.

Thus, out of 8 burners in the bottom heating zone, usually not more than 6 burners are in operation, while at the same time the four side burners in the upper zone of heating cannot provide the necessary heat; hence, on an intensified operation of the furnace, the temperature in the upper zone of heating falls by 100°C or more and does not attain the required value of 1300-1320°C. Therefore, furnaces No. 1 and No. 2 have not reached the productive capacity of 45 tons per hour as provided for in the design (at present the productive capacity of the furnaces is about 20% below the design value).

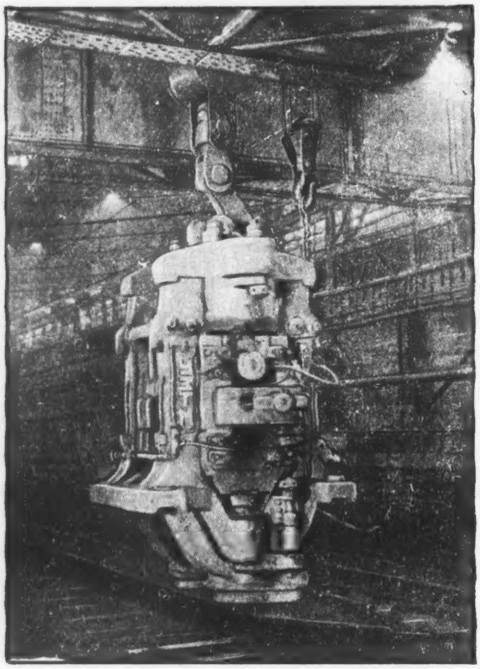
It is impossible to increase the heating capacity of the upper zone by increasing the number of burners (adding one burner to each side of the furnace) because there is no more room. Therefore work is now being carried out on the conversion of the upper zone burners to a bigger heating capacity by means of increasing the dimensions of the gas nozzle and increasing the main dimensions of the burner.

CHANGE OF ROLLS INCLUDING COMPLETE HOUSING AT THE RAIL-STRUCTURE MILL

V. V. Skakun

Deputy Head of the rail-structural mill of the Nizhne-Tagil Metallurgical Combine

A fairly large number of sections is rolled on the modern large-section and rail-structural mills. The switch-over from one section to another involves considerable time losses on roll changing. In 1955, the change of rolls with complete housing was adopted for the line of the 800 mill at the rail-structural plant of the NTMK (Nizhne-Tagil Metallurgical Combine) and it resulted in a considerable reduction in the time required for this operation.



Assembled housing on the way to the stand.

The 800 mill consists of three stands – two three-high and one two-high stands – placed in tandem. The three-high stands are driven by an electric motor of 6200 hp and of 0–70–140 rpm. The two-high finishing stand is driven by a motor of 2500 hp and 0–80–160 rpm.

The housings of both stands are open-topped with the top firmly attached to the housing by means of a key fastening. The mill is equipped with manipulators and tilters of an original design which permit a simultaneous rolling of three or four pieces. The mill is employed for the production of R-38, R-43 and R-50 rails, ordinary and light-section beams and girders, and sevrral sections for railroad cars, tractors and automobiles.

The housings for each section to be rolled are assembled on a platform which consists of plates supported on foundations, and spare housings are placed on these plates (see figure).

The platform is equipped so that the housings with the rolls can be fully assembled and adjusted. All work on the stand is done by the roll changing team under the supervision of a senior rolling mill operator.

The platform for the spare housings and rolls for the 800 mill has two substantial disadvantages: a) the platform plates are set level with the floor of the plant without being sunk and hence the work on assembling and adjusting the rolls is rather difficult because the platform for the workers servicing the housings is 1.5 m above the level of the floor and cannot be made sufficiently wide owing to the size of the rolling field; b) the plates of the stand are set at 90° to the foundation plates of the mill. Consequently, the roll housing weighing about 100 tons and suspended on the crane hook either has to be turned manually or it is necessary to use as a support for this purpose the spindles of the 800 mill or the spare housing on the stand. Extra time is lost on this operation.

For the purpose of reducing the time necessary for roll changing, schedules of the sequence of operations on the change-over from one section to another incorporating a rational utilization of the cranes, were worked out.

The foreman and senior operator of the shift inspect the assembled housings and rolls on the stand and check the assembly of the upper roll and of the top of the housing of the two-high stand (the rolls alone are changed on this stand). The operators check the tools required (wrenches, hammers, crowbars) and prepare torches and wire. Before the roll change, the crane operators check if the crane is in working order.

15 workers are engaged on roll change; one senior operator, 4 operators, one assistant operator, 6 workers of the mill, 2 crane operators and one plumber. The workers dismantle the top of the finishing stand, take off and put on the couplings, and set the springs of the two-high stand. The mill operators dismantle the old housing and take off the lower roll of the two-high stand; they take down the plates from the lifting tables, remove the keys and fastenings of the three-high stands to the foundation plates, insert new rolls and housing in the two-high stand. During the roll change the plumber disconnects, and then connects the water supply for cooling the necks and the passes. General supervision on roll changing is the responsibility of the senior mill operator and the foreman of the mill.

The most arduous operation proved to be the large-unit roll change on the 900 mill. The adoption of the roll change with complete housing on the 800 mill and the large-unit roll change on the 900 mill made it possible to reduce the time of the roll change from 3.5 hours in 1954 to 1.8 hours in 1957.

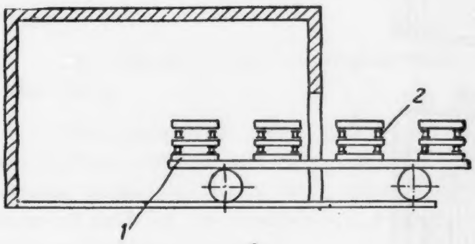
To speed up the roll changing and to facilitate assembling the housings, it is necessary to set the foundation plates of the platform for the spare housings for the 800 mill parallel to the line of the foundation plates of the 800 mill and to lower them by 1.5 m. In addition, in order to reduce the time required for the adjustment of the equipment and rolls of the finishing stand, it is necessary to introduce the change of rolls with complete housing on the two-high stand also.

MODIFICATION OF THE METHOD OF CHARGING METAL INTO FURNACE

V. P. Emel'ianov

The Magnitogorsk Metallurgical Combine

In accordance with the established procedure, part of the plates at the thick-plate mill is rolled in two stages: heating and rolling ingots into slabs, and rolling slabs into plates after a second heating. Before being rolled, the slabs are heated in removable hearth furnaces (Fig. a).



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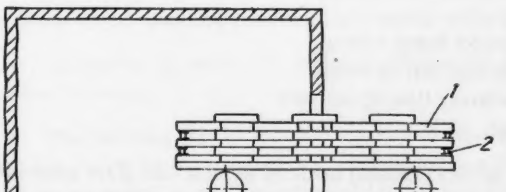


b

Charging slabs into furnace:
a) old method; b) new method; 1) slab; 2) support.

Previously, the slabs were charged into the furnace in separate piles, metal supports being placed between the slabs to separate them from each other. Asbestos sheets were inserted between the support and the slab to prevent sticking. With such a method of stacking the slabs, the furnace was not fully utilized and the mill used to stay idle because of the shortage of heated-up slabs. At the same time large amounts of asbestos sheets and metal supports were consumed owing to fusion in the course of heating.

With the object of speeding up the heating operation and reducing the consumption of supports and asbestos, as well as easing the work of furnace operators, the foreman G. V. Konovalov and N. M. Shemetov proposed a new method of charging slabs on the removable furnace hearth.

According to the new method (Fig. b), the slabs are stacked in such a way that each row overlaps the gaps between the slabs of the previous row. The overlapping amounts to 50-80 mm. The changing of the slabs by staggering them ensures the rigidity of the slab pile on the hearth, speeds up the charging operation

and increases the output of the furnace by a factor of 1.5.

The elimination of operations on placing the supports and asbestos plates, which are now used only for the slabs at the end of the pile makes the work of furnace operators considerably easier.

THE APPLICATION OF BUFFERS IN PLACE OF SUSPENSION GUIDES

I. T. Lyzhenko

Senior foreman of the 800 mill at the Chusova Metallurgical Works

The entry table, installed on the upper rest bars at the rear end of the second (finishing) working stand of the 800 mill, left no room for the hanging guides in the five bottom passes. If, for any reason, the rolled piece was bent upwards on leaving the pass, it dislodged the upper rest bar together with the roll fittings mounted on it. This resulted in incompletely rolled pieces, breakage of the rest blocks and mill stoppages.

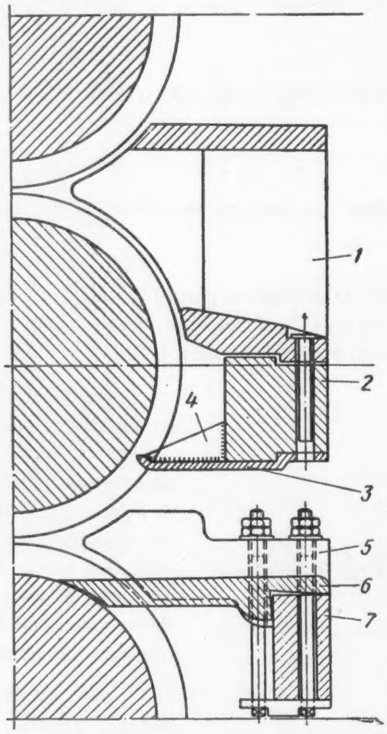
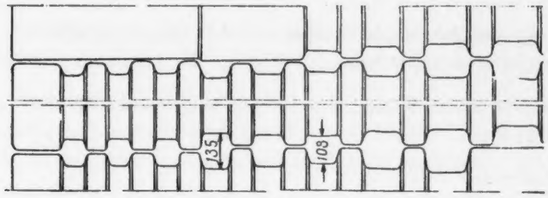


Fig. 1. Roll accessories for beam rolling; 1) entry table; 2) lower rest bar; 3) buffer; 4) buffer support; 5) delivery line; 6) delivery guide; 7) lower rest bar.

To prevent the dislodging of the upper roll fittings, buffers (fenders) made of 20 mm iron plate were welded onto the lower side of the upper rest bar above each of the four passes (see figure). The length of the buffers is such as to permit the setting of the rest bar into the recesses of the housing in the presence of the rolls in the stand. This simple arrangement completely eliminated the dislodging of the upper roll fittings on the rear end of the second working stand. The breakages of the rest bars ceased, there was no more defective material due to unfinished rolling, and the stoppages of the mill decreased.

RATIONAL UTILIZATION OF THE UPPER ROLL BARREL

For a long time, only one half of the upper roll barrel (Fig. a) in the finishing stand of the 800 mill at our Works was utilized. With the object of cutting down roll consumption, exactly the same grooves as on the first half of the roll were cut on the other half (Fig. b). In addition, the positioning of the grooves on the middle and the lower rolls was slightly modified by changing the dimensions of the collars.



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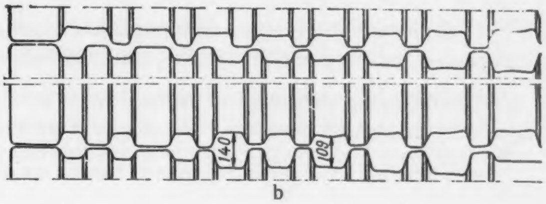


b

Thus, now when the rolls are changed, only the lower and the middle rolls are replaced, and the upper roll is taken out, turned around and placed back on the stand.

As a result of the rational utilization of the upper roll barrel, the consumption of rolls decreased markedly.

I. T. Lyzhenko

EXPERIENCE ON THE OPERATION OF THE AUTOMATIC TUBE MILL 400

I. G. Mindlin

Deputy Head of the tube plant at the Zakavkaz Metallurgical Works

Tube production at our Works constitutes about 50% of the total output of all the rolling plants of the Works.

The tube-rolling unit 400 which includes the automatic mill is the first tube plant designed and made in the Soviet Union.

The plant consists of two circular rotary hearth furnaces for heating tube billets of 170-350 mm diameter, two piercing mills with rolls of a variable angle of inclination to the axis of piercing, an automatic mill with preceding reheating furnace, two sizing mills, one seven-stand hot-rolling mill (with a separate drive for each stand), two seven-roll mills for tube straightening, a mechanized tube-cutting and tube-threading machine, inspection benches and other auxiliary equipment.

All d.c. electric motors of the main drives are enclosed and could therefore be placed directly at the mills. The power to the piercing, automatic and sizing mills is supplied through mercury arc rectifiers.

Except for the automatic mill, all other mills are provided with roller bearings (the automatic mill has textolite bearings).

The delivery side of the piercing and the sizing mills is provided with adjusting and controlling roller centering guides for the plug and the pierced shell, and with electric motors for the supporting bearings of the plugs which are mounted on those bearings and move together with them.

For the first time on the automatic mill a mechanism for plug changing, a crankgear for fitting the upper roll and a mechanism for turning the tubes after the first pass were applied. An underground conveyer is employed for the removal of cuttings and chips from the tube-cutting machines. An automatic centralized lubrication system is extensively applied at the mills.

The availability of two piercing mills makes it possible to employ one or two consecutive piercing operations in the production of tubes.

At present, all tubes except those of minimum diameter (168 mm) are produced in twq piercing operations and therefore tube rounds of small diameter can be employed.

Since the unit 400 was put into operation (December, 1953) the production of 168 mm to 325 mm diameter tubes for the petroleum and coal industries, the machine building industry, agriculture, power stations and civil engineering etc. has been mastered. About 10% of the production is for export.

In the course of operation, advantages and disadvantages of the unit have been found so that the technological process, rolling equipment and separate mechanisms could be improved.

The following technology of production has been adopted at our Works. Steel is bottom-poured into square, big-end-up molds with a hot top. The weight of the ingot is 6 t.

The dimensions of tubes, rounds and blooms are as follows:

External diameter of the tube	168	219	245	273	325
Diameter of the tube round	170	180	230	250	270
Cross section of the bloom	255 x 310	255 x 310	255 x 310	280 x 310	300 x 320

The 255 x 310 mm blooms (for 168 mm-245 mm diameter tubes) are subjected to pneumatic chipping or torch scarfing while cold, then they are heated in holding furnaces and rolled to tube rounds on a bar mill and again conditioned in the second finishing shop.

The 280 x 310 mm and 300 x 320 mm blooms (for 273 mm and 325 mm diameter tubes respectively) are rolled into rounds without being heated and therefore their surface defects cannot be removed. Hence, the quality of the 250 mm and 270 mm tube rounds is somewhat lower than that of the 170 mm to 230 mm rounds.

With the object of improving the quality of the tube rounds, especially for large-diameter tubes, the finishing plant of the bar mill will be modified according to the design by the Gipromez, and the "brightening" of rounds will be introduced. The design should incorporate the installation of modern machines for continuous brightening of the rounds over the whole surface and not only a partial brightening (on a spiral path) as envisaged under the existing technical conditions. This is especially necessary in connection with the increase in the output of large-diameter, thin-walled tubes.

The tube rounds are heated in one of the two circular furnaces (when one furnace undergoes major overhaul, the other is put into operation). At present, preparations are being made for the change-over to heating the rounds for large diameter tubes in two furnaces simultaneously.

The rotary hearth with concentric work supports (ribs) proved to be impractical because the gaps between the supports filled up relatively fast with scale and the supports themselves deteriorated. The change-over to a rammed hearth (70% chromite, 15% chamotte powder and 15% scale on liquid glass) based on the practice of the Novotrubnoi Works was found fully justified.

Important technical improvements on the unit 400 are the introduction of the centering of hot tube rounds and the installation of "permanent" (long lasting) hollow mandrels with internal cooling.

For the centering of the front end of the tube round, the pneumatic gun of N. A. Andreev design was constructed and placed above the roller table conveying the rounds from the rotary hearth furnaces to the piercing mills. The necessary indentation in the form of a conical depression 30-40 mm in diameter and 30-35 mm deep on the front end of the round is produced by a single blow of the striking pin. The gun is situated above the roller table and, therefore, after each "firing" its barrel is lifted to allow the centered round to pass to the piercing mill, and then the barrel is again lowered into the working position. The gun is operated automatically. The centering of the front end of the rounds facilitates the piercing process and results in a better uniformity of tube wall thickness.

In the second half of 1957, experiments were carried out and satisfactory results were obtained on piercing large batches of rounds 180 mm-270 mm in diameter and 1600 mm-3500 mm long of steel 10, 20, D, with hollow mandrels rigidly connected by a threaded joint to the end of the mandrel support bar.

The mandrel was internally cooled with water under 10-15 atmos pressure through the ordinary fixed pouring tubing of the support bar and for this purpose a centrifugal pump was installed at the piercing mill. In addition, during the usual break after each piercing operation, the mandrel was intensibly cooled from outside by means of a spraying equipment (when the mandrel support bar was returned to its rear position).

The life of the mandrels was very effectively increased owing to the 4 mm - 5 mm diameter holes drilled in front of the pointed end of the mandrel. The "permanent" mandrel lasts for 400-800 piercing operations, i.e., 10-20 times more than ordinary mandrels. Moreover, the mandrel efficiently centers the tube round and so improves the uniformity in the wall thickness of tubes.

The production of thin-walled, large-diameter pipes (168 × 6 mm and 325 × 8 mm) on the automated unit 400 was mastered. In 1957, the production of those pipes constituted 15% of the total output.

The process of rolling the tubes on the automatic mill has its pecularities: it is a process of longitudinal rolling in a circular pass and it is well known that this process involves varying-magnitude stresses of separate elements of the section rolled because of nonuniform drafts and variable peripheral velocities of the rolling surfaces of the roll grooves.

The most dangerous tensile stresses appear in the second pass when the tube, turned through 90°, does not fit tightly to the walls of the pass in the sections near the delivery side; during the rolling of thin-wall tubes this results in characteristic breaks in form of crescent-shaped fractures of the tube walls.

In addition, thin-walled tubes cool at a fast rate and hence the plasticity of metal in the course of the complex processing on several mills is lowered.

In the production of thin-walled tubes, the removal of external defects is also more difficult owing to the reduction in the absolute values of the tolerances in the wall thickness. Hence while economic advantages (the reduction in tube weight by 10-15%) in the production of thin-walled tubes are evident for the consumer, at the same time disadvantages for the producer have been observed: metal consumption increased by 5% and power consumption by 10-15%; the output of I-grade tubes decreased and the output of the II-grade and 2nd sort tubes increased correspondingly; the productivity in a busy time fell by 15-20% and as a result of increased pressures and frequent replacements of equipment, necessary to obtain more accurate dimensions of the tubes, the consumption of the equipment on the automatic, the rolling and the sizing mills increased.

The following measures could be conducive to a higher output of thin-walled tubes and to an improvement in the technical and economic indices of production: 1) an improvement in the quality of the initial material and the reconstruction of the finishing shop of the bar mill, including the introduction of brightening of the tube rounds; 2) the acquisition, installation and use of a new 1500 ton horizontal press for breaking large-diameter tube rounds with a minimum tolerance for tubes of every diameter; 3) the use of two rotary hearth furnaces for heating the rounds for the production of large-diameter tubes; 4) the adoption of "permanent" mandrels in the first piercing operation; 5) an improvement in the quality and, later on, a reduction in the ovalness to 0.3-0.4 mm of the mandrels of the automatic mill; 6) testing and, in case of satisfactory results, the introduction of a new mechanism for mandrel changing to replace the present one whose slow operation results in an excessive cooling of tubes.

In conclusion, it must be mentioned that the production of the tube plant 400 is not yet completely realized (in the assortment of tubes produced at the plant, the amount of tubes 168 mm in diameter reached 54% in the second half of 1957). The tube plant 400 must be completely relieved of the production of 168 mm diameter tubes and the production of these tubes must be transferred to the tube plant 250 at the Baku Tube Works, this move being in line with the policy of specialized tube production.

THE CONFERENCE ON THE PRODUCTION OF THIN-WALLED TUBES

From February 25 to 27, 1958, the Conference of Tube Industry Workers on the Production of Thin-Walled Tubes was held in Dnepropetrovsk. Workers of tube, iron and steel, and machine works, of the Gosplan (State Planning) of the USSR, Russian FSSR and Ukrainian SSR, of the National Economic Councils, of the Scientific Research and design institutes, higher educational establishments and technical schools, representatives of scientific and technical journals, Scientific and Technical Department of Ferrous Metallurgy, and Party, Trade Union and government officials took part in the Conference. In all, 485 men, representing 75 organizations, took part in the Conference.

In the course of an extensive discussion on the papers and reports presented at the Conference by the Institutes, the Gosplan of the Russian FSSR, tube works, and iron and steel works, it was noted that in spite of an increase in the output of thin-walled seamless and electrowelded tubes in recent years and the adoption and expansion of the production of extra-thin-walled stainless tubes, thin-walled tubes for gas pipelines and other tubes, the requirements in tubes for the national industries are not fully met. This situation is due mainly to the absence of serious technical and economic research in the field of thin-walled tube production, the shortcomings in tube production planning, delays in designing new equipment for the tube mills, equipment for finishing operations and thermal treatment of tubes and instruments for the automatic control of the dimensions and quality of the tubes, and a low quality of tube mill equipment.

The Conference recognized the urgency of carrying out scientific research work on thin-walled tube production (the All-Union Scientific and Research Tube Institute will coordinate this work), to organize the production of thin-walled tubes on 250 and 400 pneumatic mills, on butt-welding mills and other mills, and to develop a new system of planning tube production.

The Conference considered that in the meanwhile, before the results of technical and economic research are known, it would be desirable to put into production a new assortment of seamless, welded and cast tubes taking into account various methods of production; the conference advised the Committee for Standards and the VNITI* to revise in the course of 1958-1960 the existing standards for seamless, welded and cast tubes. The resolutions of the Conference include concrete recommendations for tube works and National Economic Councils with regard to thin-walled tube production.

The Conference was well prepared and was conducted in a business-like and creative atmosphere.

* VNITI – All-Union Scientific and Technical Institute.

MISCELLANEOUS PRODUCTS

EQUIPMENT FOR THE MANUFACTURE OF PERMANENTLY TWISTED STEEL ROPE

Engineer B. M. Vorontsov

The most durable steel ropes in service are permanently-twisted wire ropes. They are made by means of deforming their strands on a special machine.

In the process of drawing and twisting a steel wire into a rope, stresses appear in the wire and have a detrimental effect on the serviceability of the rope. In order to increase the durability of the rope in service, it is necessary to remove these stresses, the rope remaining then permanently twisted. The rope strands are given a spiral form and they should be twisted in that form on the finished rope.

The ends of the permanently twisted ropes are easier to fit into couplings or end pieces; they are easily assembled as they do not tend to form kinks.

The attachment, used at the Leningrad Steel-Rolling Works, for the manufacture of permanently twisted rope is shown in Fig. 1. It consists of six three-roll sections (roll diameter = 20 mm) positioned symmetrically on a circle, at an angle of 24° to the shaft of the machine; the rolls revolve on their axes when the strands pass over them. The distance between the external rolls is fixed (43 mm + 43 mm) and the middle rolls can move on a circular path through a definite angle, depending on the size of the strand being deformed. The rolls are mounted on a single disc which can be rotated by means of a worm gear. Such a design makes it possible to displace all the middle rolls simultaneously through an equal distance with respect to the external rolls.

When the strands pass over the rolls, forces appear which tend to displace the middle rolls and to upset the correct process of strand deformation. But the worm gear keeps the rolls in the predetermined position.

A graduated scale is provided for a very accurate adjustment of the disc with the middle rolls.

The strands bend around the rolls, as shown in Fig. 2, and pass to the clamping blocks where they are twisted into the rope when the machine revolves together with the attachment.

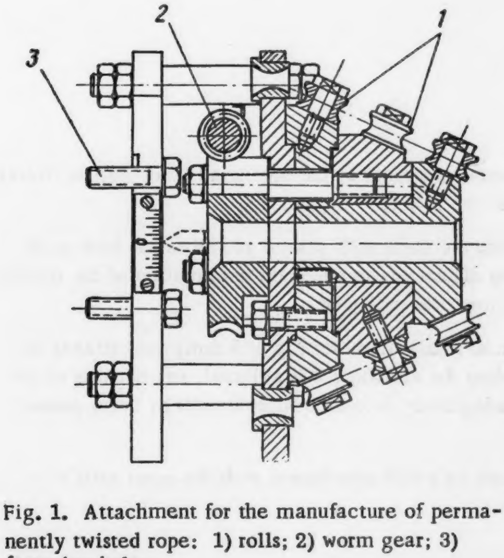


Fig. 1. Attachment for the manufacture of permanently twisted rope: 1) rolls; 2) worm gear; 3) fastening bolts.

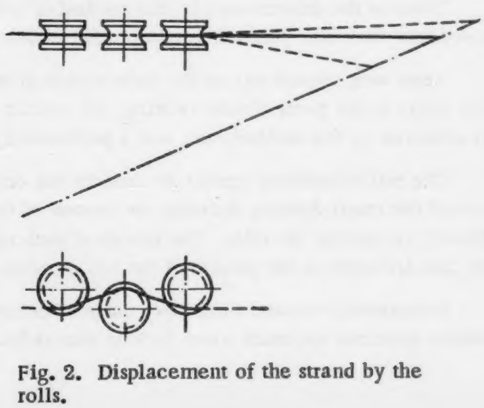


Fig. 2. Displacement of the strand by the rolls.

In order to achieve the correct deformation of strands, it is necessary to set the gripping blocks at a definite distance from the axis of the nearest roll. The positioning of the outside rolls is chosen depending on the mean value of the winding pitch of the rope produced on the machine.

The inclination angle of the strands from the outer roll to the gripping blocks will vary depending on the given pitch of twisting the rope. Ats a small pitch the blocks have to be positioned near the strand deformation attachment; on leaving the outer roll, the strands will be bent unnecessarily. To avoid this bending, a deformation equipment of a smaller size must be employed.

The attachment is suitable for rope of certain diameters and in particular for the 6/400 basket-type rope-twisting machine. Two deformation attachments were made: one for ropes 6 mm to 10 mm in diameter and the second for ropes 11 mm to 15.5 mm in diameter.

David Bagradze (on the left), senior rolling-mill operator, and N. Borzov, foreman of the 800 mill at the rail-structural plant of the Azovstal Works, – both winners in the competition for the first half of 1958.

When the deformation attachments are used, it is possible to effect the deformation of the strands with three rolls positioned in one line without the displacement of the middle roll, the middle roll being situated at an equal distance from both outer rolls. The distance between the two outer rolls is then calculated from the formula

$$l = 3d + 2D,$$

where l is the distance between the outer rolls;
d is the roll diameter;
D is the rope diameter.

Tests on the deformation by this method of a small amount of strands at the Leningrad Steel-Rolling Works showed that the above formula is suitable for twisting ropes of the $6 \times 19 \times 1$ type.

Tests were carried out on the deformation of strands, with the outer rolls placed at a distance from each other equal to the pitch of robe twisting, the middle roll being displaced; an adequate deformation of the strands was achieved by this method also, and a permanently twisted rope was produced.

The roll attachment cannot be used for the deformation of small-diameter (up to 5 mm) steel strands because of the small distance between the centers of the rolls when the attachment is adjusted, and because of the difficulty in spacing the rolls. The strands of such ropes are adequately deformed without rolls by being passed over pins arranged on the pattern of the roll attachment.

Permanently-twisted steel ropes made of deformed strands on a roll attachment with the outer rolls at a constant distance, are much more durable than ordinary ropes.

Tests on 15.5 mm diameter rope (GOST 3070-55) of the $6 \times 19 \times 1$ type showed that it is advantageous to use ropes made of deformed strand for a pulley where the ratio $\frac{D}{d} \geq 20$ (d = rope diameter, D = pulley diameter). This ratio is usually chosen for crane lifting ropes. Under those conditions the life of permanently twisted ropes is approximately 1.5 to 1.7 times longer than that of ordinary ropes.

ORGANIZATION OF PRODUCTION QUALITY CONTROL

In the article "Organization of production quality control" by N. P. Inozemtsev, Ia. I. Sokol, I. F. Rysev, D. A. Tarasenko and S. I. Zamiatin, published in the "Metallurgist" No. 9, 1957, the problem of OTK staff reduction was raised. We publish below a reply to that article.

THE OTK CONTROLLER IS A PRODUCTIVE WORKER AT THE PLANT

M. P. Popov

Head of the OTK at the Iakubovskii Works

In the last two years, the majority of works have made considerable reductions in their OTK staff. In this connection the role and responsibilities of foremen, shift and section heads and plant heads for the quality of production have increased. At the tube plant of our Works there are three old stationary tube-welding machines, including finishing sections, in operation. Each machine operates on the principle of a continuous production line; all tubes are inspected. Hardly any unfinished product is left at the end of each shift.

The adoption of a continuous production line resulted in a more efficiently planned working day of OTK controllers. In addition to controlling duties, they perform productive work. This does not affect the efficiency of inspection. The Works has received no complaints from tube consumers.

In each shift, at each machine, there is one controller who inspects the tube exterior, measures the tubes and records how many tubes each welder produces (four welders are employed at each machine). With the object of reducing defective product and metal consumption, the mill operator stamps the number of the respective welder on each tube, 100 mm from its front end, before the tube is entered into the machine. At the receiving end, the OTK controller inspects the tubes and marks inefficiently welded ends and defective spots with chalk, and passes the tubes for cutting. The OTK staff takes samples and carries out cone tests and bending tests. By means of these tests, defects can be revealed and the production of inferior tubes prevented. The percentage of defective tubes in June this year fell to 0.42% compared with 0.7-0.8% for the period from January to May.

Owing to OTK work being organized in this way, the amount of unfinished production decreased considerably. On the first of each month there is about 5 to 10 tons of unfinished product. At the Vyksa, the Leningrad, the Lenin and the Andreev Works, the hydraulic testing is carried out by plant workers, the OTK controller merely supervising the testing. At our Works, however, the hydraulic testing machines are operated by OTK controllers; they sort the tubes, separate out rejected tubes, keep records of defective and rejected tubes produced by each welder, give signals if unacceptable tubes are produced and request the foreman to improve welding. An OTK controller brings the tubes to the testing machine.

At the final inspection of tubes, OTK workers inspect the tube surface, check the quality of the threading, mark defective places for second cropping with chalk, throw defective tubes into receptables, place accepted tubes on shelves, align the ends with the flange, lubricate the thread on the tubes and flanges, and screw the flanges onto the tubes. Two men are engaged at the acceptance station for tubes.

At the old tube-welding plant of the Andreev Works, in addition to six OTK controllers there are six workers in each shift engaged in screwing flanges. The same number of controllers is employed at the Vyksa, the Dnepropetrovsk and the Lenin Works.

All workers in the tube plant at our Works are paid according to the amount (length) of tubes produced. Therefore all personnel, from the welder to heads of plant, shifts and teams, strives for the economic utilization of metal, aiming at the greatest possible length of tubes. At present the plant produces tubes 6.7-6.9 m in length, on the average, from strips 6.7 m in length. From strips of the same length, tubes 6.5 m long are produced at other Soviet works.

The length of the tube depends to a great extent on the cylindrical section of the welding bell. The bells at our Works have the smallest dimensions. As a result of the small diameter of the cylindrical part of the bell, the elongation of the strips amounts to 12%, while at other works it is 7-8%. Owing to this fact alone, we add 200-300 mm length to each tube.

As a result of the above measures, the workers exceed the State-planned output by 4-6% daily, save metal and reduce defective production. In June, the tube output at the plant amounted to 120.9% of the planned target.

FROM THE HISTORY OF TECHNOLOGY

THE DEVELOPMENT OF THE FORGING INDUSTRY

Of all the various forms of metal working under pressure, the oldest method was free forging in which the shape and the size of the billet changed under the impact or pressure of the hammer. This process passed through several development stages. When man learned to produce copper from ore, the forging of metal on an anvil or a wooden block began to be practiced for the purpose of working the cutting edges of cast tools or manufacturing useful articles from copper and bronze plate.

The forging process has occupied one of the most important places in the iron industry. Initially (before the XIIIth century), all forging operations were carried out manually. Productive efficiency was extremely low. For instance, to forge 1 pood (16 kg) sheet billet from a bloom approximately 13 man-hours were required.

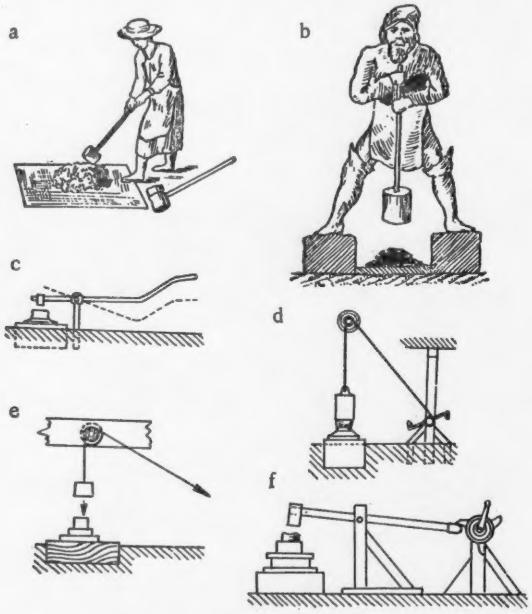
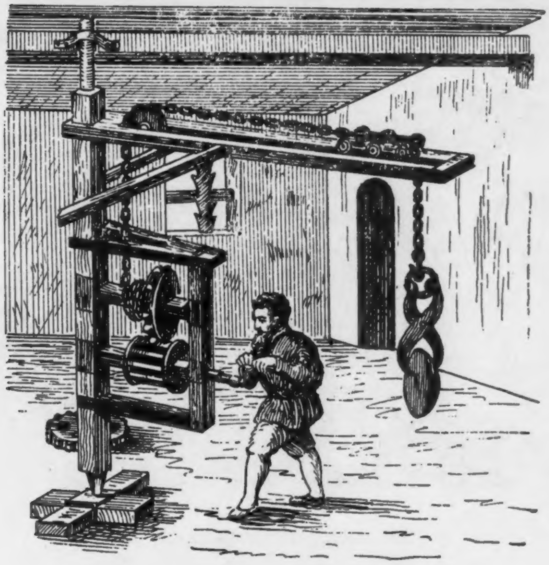


Fig. 1. Development of manually operated sledge and drop hammers:
a and b) sledge and drop hammers, operated directly by hand; c and d) simple hammers actuated indirectly by hand; e) drop hammer with a windlass; f) sledge hammer driven by a hand wheel.

As the weight of forged pieces increased, manual working became difficult. This was conducive to the development of sledge and drop hammers (Fig. 1). A slow-speed sledge hammer of 30 strokes per minute and with 75 kg dropping weight replaced 25-30 workers. Even then, however, metal working was time-consuming, since the impact force effected by manually operated equipment was very limited.



Rotary jib crane.

Forging an anchor manually.

In the XIIIth century, the water wheel began to be used as the motive force for forging hammers. Thus began the second stage of the forging industry characterized by the mechanization of the main operations (all auxiliary operations were carried out manually as before). Much less time was expended on bloom and billet working.

In the XVIth century, iron works equipped with water wheel driven hammers came into existance. The bloom was removed from the bloomery by means of crowbars and tongs. Two assistants hammered it with big wooden hammers and then cut it into 4 or 6 pieces with a heavy hammer and a chisel. The pieces were heated again and forged into sheets, rods and other products (wheel rims, plowshares, etc.)

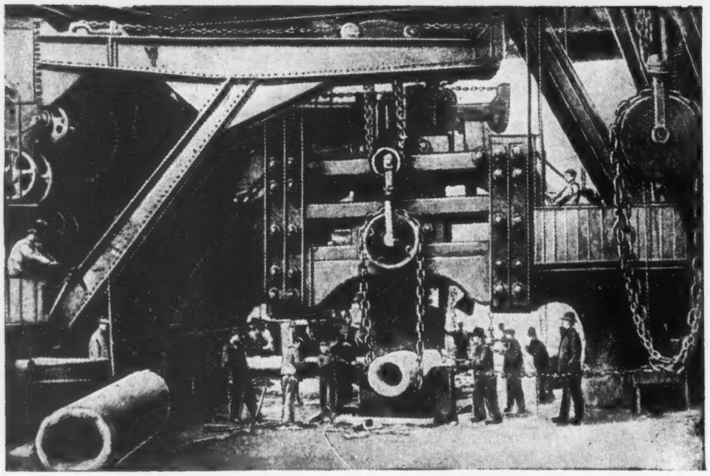


Fig. 4. Forge shop at the Krupp Works in Essen.

The bloom was usually 50-70 kg in weight. In six working days, a foreman and two assistants would forge 200 to 300 kg iron products from blooms. Heavy blooms were removed from the furnace by means of large tongs suspended by a chain on a rotary jib crane (Fig. 2). The increase in bloom weight necessitated an increase in the weight of the hammer head. The weight of the dropping parts of hammers amounted in some cases to 490 kg, and cranes were extensively employed for lifting and transferring blooms and billets from the furnace to the anvil. These rotary cantilever cranes were essentially similar in design and operation to the cranes employed in modern forge shops.

In the XVIth century, tilting hammers became very popular. They were used for forging thin sheet iron.

Subsequently, tilting hammers were employed for stretching wire, making gun barrels, scythes, knives, etc. The weight of the dropping parts of those hammers did not exceed 60-80 kg.

In the XVIIth century, the most difficult pieces for forging were anchors, anvils and rolls for rolling (flatting) mills. The weight of the anchor depends on the ship tonnage (displacement); if a 20 t ship required an anchor of about 44 kg weight, the anchor for a 1500 t ship had to be more than 3 t in weight. Almost till the end of the XVIIth century, anchors were forged with hand hammers (Fig. 3) but later on water-driven hammers were introduced (France). The first report on the production of anchors by means of water-driven hammers was made in 1723 by Reaumur in the Academy of Science in Paris. The manufacture of an anchor consisting of a shank, arms, two flukes and a ring was an arduous and costly operation; it was necessary to make each part separately and then weld them together. The heavy billet was moved by a stationary crane which served the forge and the hammer. For welding separate parts together, two cranes and two forges, where the ends of the parts were heated, were necessary.

Expecially heavy blows were required to weld the arms of the anchor to its shank and for this purpose some forge shops had hammers with a large dropping height of the hammer head. 4-5 blows were necessary in a welding operation. In the forge shops where water power could not be utilized, other machinery was employed for this work, as, for instance, drop hammers similar to a pile driver (the hammer head was lifted with a rope, pulled by 7 or 8 men), or suspended bars, weighing about 120 kg, with sharpened ends for a better penetration of the weld. One worker guided the bar and 7 or 8 other workers hoisted it with a rope.

Steam engines came to play an important part in the XVIIIth century industry. Axles, wheels, drive gears and shafts made of wood were replaced by cast and forged iron parts. The variety of forgings increased and so did the weight of the forged parts. The water-driven hammers were inadequate in the manufacture of large forgings and inventors began to work on the adoption of the steam engine for iron forging. Initially, in puddled iron production, hot blooms were forged by cast iron hammers whose dropping parts weighed 600 kg and which were driven by a steam engine.

According to Johansen, the bloom was welded to a special handle by which it was held, and was forged into an ingot 50 × 7 × 10 cm. The operation took 7-8 minutes. One hammer served 12 puddling furnaces.

The invention of the steamship, the locomotive and the development of railroads in the beginning of the XIXth century resulted in a new market for iron products. The use of plate iron in the shipbuilding industry called for facilities for plate cutting and rivet hole punching. In the beginning of the XIXth century, several designs of alligator type shears, used in puddling iron production and for cutting rolled metal, were developed in England. At the same time, the first punching machines and special machines for rivet manufacture appeared. These machines were steam driven in England and water driven on the Continent.

Already Watt had expressed an idea of joining the hammer head directly with the engine shaft (English patent, 1784). The first steam hammer was built to Nesmit's design in 1842, for the Krezo works.

With the introduction of the steam hammer, the third stage in the development of the forging industry began. The forging operation became simplified and the weight of forgings increased considerably. Large forgings were not welded from separate parts but were forged in one piece. A substantially smaller number of workers was required, the time of the forging operation was considerably reduced and the layout of forge shops underwent a change.

The weight of the drop head of the first steam hammers for rough forging was 1.5 to 2 tons. In 1893, the Bethlehem Works (U.S.A.) had a 100 ton cast ingot for armor plate at the Chicago Exhibition. For forging such ingots, a steam hammer whose drop head weighed 125 tons was installed at the Bethlehem Works; the drop head was lifted to a height of 6 m and the anvil weighed 2150 tons. Figure 4 shows the forge shop of the Krupp Works with a 50 ton hammer served by a steam crane provided with an iron girder and turning equipment. On the proposition of Gladhill, the manager of the Whitworth Works in Manchester, the first hydraulic press for forging iron was built in 1860-1861. At about the same time (1861) John Gaswell, an Englishman, built a hydraulic forging press in the State Railway Works in Vienna.

The advantage of the forging press over the hammer was recognized when huge steel ingots for guns, steam ship shafts and other large equipment, had to be forged, since the weight of the hammer had to be at least equal to the weight of the forged piece otherwise it was impossible to change the shape of the piece.

To forge a 15 cm gun from a 36 ton ingot with a 50 ton steam hammer, three weeks were required (33 reheatings) while a 4000 ton hydraulic press completed such an operation in four days and needed only 15 reheatings. Thanks to the smooth action of the press, all attachments (tools, dies, blocks etc.) could be made of cast iron.

The growth of machine production towards the end of the XIXth century necessitated an improvement in forging techniques and the replacement of manual operations on metal working by machine processes. At that period, a number of forge-shop tools such as drifts, dies and various devices for holding and turning the forgings appeared.

About 1860, hydraulic cranes (bridge, gantry and travelling cranes) for transporting billets and forgings, and, in 1870, various hydraulically operated travelling trolleys (tables) and roller tables were introduced. At the same time, hydraulic shears operated by one worker were put to use for cutting rods and rails.

In Russia, the first steam crane for local production needs was built at the Perm Steel Works in 1863.

The introduction of electrically-driven lifting and transporting machines made the design and operation of crane mechanisms much more simple. In 1885, Siemens constructed an electric revolving crane for a London factory. In time, steam was completely replaced by electricity. In 1882, in England, Professor Jenkin proposed a system of aerial transportation, called telpherage. Later on – first in the U.S.A. and then in West Europe – an aerial electric transportation system of special design was introduced on an extensive scale for internal transport needs at works.

In the 90's of the XIXth century, the first patents for hydraulically driven manipulators were taken out in Germany. They began to be widely used about the middle of the XXth century.

In this way, towards the end of the XIXth century, all the essential means of mechanizing forging production were already put to use in industry. These means underwent a further improvement and development in the XXth century.

NEW BOOKS

G. M. Malakhov, N. I. Starikov and A. G. Shostak

THE MAIN IRON ORE BASE OF THE USSR

Moscow, Metallurgy Press, 1957, 162 pages

The book contains an account of the development of the Krivoi Rog Iron Ore Basin. The authors give the history of prospecting and mining the deposits, describe the geological features of the Basin and discuss in detail the mining technology and the development of ore mining in the Krivoi Rog district till the October Revolution.

Beginning in chapter III, the authors describe, step by step, the development of the Krivoi Rog Basin during the years of Soviet rule. An account is given of the reconstruction of the Basin after the Civil War, the modernization of ore mining methods and the systems of working out the deposits as practiced by the Krivoi Rog miners. Next, the authors describe the destruction of the Krivoi Rog mines becaused by the facists aggressors, the rebuilding and the development after the Great Patriotic war, of this, the largest, iron ore base of our country. New methods and techniques employed by the Krivoi Rog miners are presented. Tables and figures fully illustrate the material presented.

The book is of interest to a wide circle of readers.

METALLURGY ABROAD

CENTRIFUGAL CASTING OF TUBES IN THE PEOPLE'S REPUBLIC OF CHINA

Cand. Tech. Sci. L. S. Konstantinov, B. D. Khokhalin and Engineer A. N. Smoliakov

At the pipe-casting works of the Anshan Metallurgical Combine, cast iron water pipes of 100-150 mm and 700-1000 mm in diameter are cast into sand molds by the vertical casting method and pipes 200-600 mm in diameter are cast by the centrifugal method.

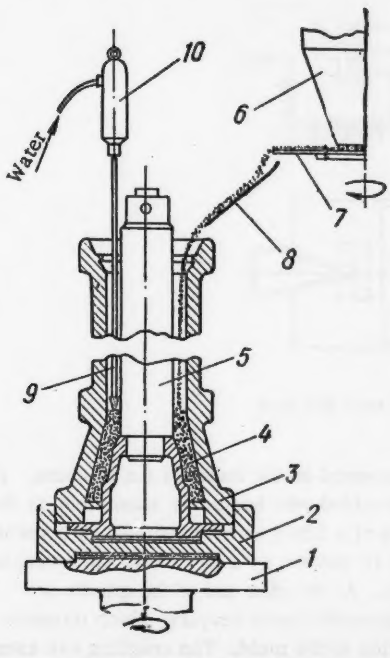


Fig. 1. Assembly of the flask for pipe casting: 1) revolving table; 2) bottom plate; 3) flask; 4) socket pattern; 5) central core pattern; 6) bin; 7) plate feeder; 8) trough; 9) ram; 10) vibrator.

The technological process of centrifugal pipe casting consists of the following operations:

1) The preparation of the mold mixture, the preparation and drying of molds; 2) the preparation of the core mixture, the preparation and drying of cores; 3) assembling of the molds; 4) pouring the metal into the molds; 5) blowing of filled molds with air; 6) removal of pipes from the molds; 7) cleaning, hydraulic testing, painting and inspection of pipes.

The mold mixture is prepared from Anshan local sand which has large spherical grains.

The molds are formed on a special apparatus (Fig. 1). The flask is mounted on a table supporting a bottom plate and rotated by an electric motor. A pattern for the socket is fixed to the bottom plate, and the cylindrical part of the mold is formed by means of the central core pattern. The mold mixture is delivered from the bin by means of a plate feeder through a trough to the flask. It is then rammed with a ram and a pneumatic vibrator, supplied with air at 5-6 atmos pressure through a rubber hose. The central core pattern has the form of an inverted cone and on its removal the surface of the mold is smoothed out.

The flange section of the pipe is made manually with a special pattern.

The flask with the mold is set in the horizontal position by means of a tilting device. The total time for the preparation of the mold (up to the drying) is 7.5 to 9 minutes for pipe 200 mm in diameter, 10 minutes for 300 mm pipe and 11 minutes for 500 mm pipe.

The finished molds are coated with graphite paint and dried in a horizontal position by flue gas from special furnaces. The dried molds are assembled and delivered to the centrifugal machines.

There are two types of centrifugal machines at the plant. 200 mm and 250 mm pipes are cast on machines equipped with spindle drive and electromagnetic coupling (Fig. 2). Two pairs of supporting rollers revolving on

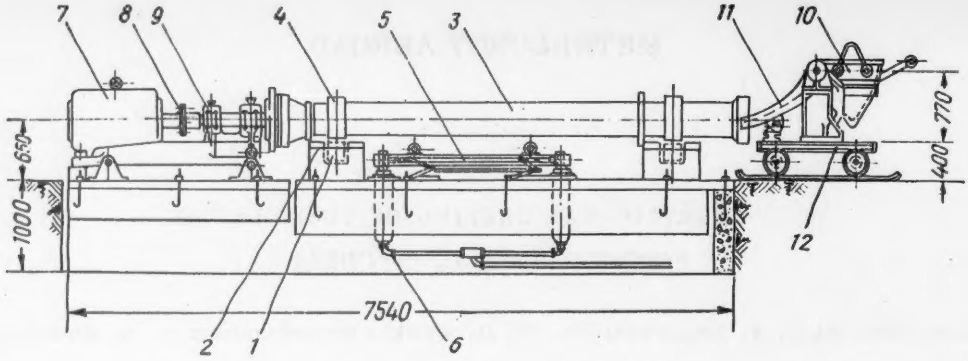


Fig. 2. Machine for centrifugal casting of 200 mm and 250 mm pipes:
1) supporting rollers; 2) bearings; 3) mold; 4) hoop; 5) lifting device; 6) hydraulic cylinder; 7) electric motor; 8) coupling; 9) spindle; 10) ladle; 11) runner; 12) trolley.

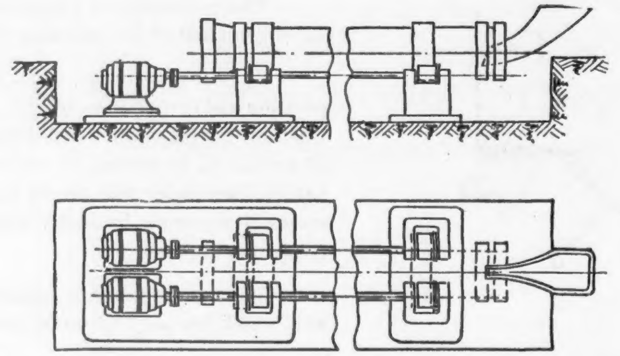


Fig. 3. Machine for centrifugal casting of 300 mm-600 mm pipes.

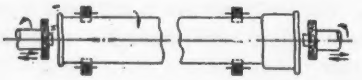


Fig. 4. Method of cleaning the internal surface of the pipe.

bearings are mounted on the frame of the machine. The filled mold, provided with hoops, is transferred on the stand by means of a lifting device actuated by hydraulic cylinders, and is rotated by a dc motor via a coupling and the spindle. At the other end of the spindle is mounted an electromagnetic coupling which transmits the rotary motion to the mold. The coupling can exert force up to 2000 kg. Metal is poured into the mold from a hinged ladle through a stationary runner on the flange side of the pipe. The ladle and the runner are mounted on a trolley. The liquid metal is mearured out by weight.

300 mm to 600 mm pipes are cast on larger machines equipped with two electric motors (Fig. 3). To avoid frequent changes of rollers when the gear ratio is changed, one motor is employed for casting 300 mm and 350 mm pipes, and the other for 400 mm to 600 mm pipes.

The times of pouring and rotating the pipes as well as the total times of the casting process for pipes of various diameter are given in the table below:

Pipe diameter, mm	*200*	*250*	*300*	*350*	*400*	*450*	*500*	*600*
Time of pouring, sec	10-14	10-14	15	15-18	15-18	16-20	18-20	18-20
Time of rotating, sec	4	4.5	5-6	6-8	6-8	10-12	12-14	12-14
Total time of casting, min	6	6.5	8	10	10	14	16	16

After the pipe is cast, the sand layer of the mold is pierced with a tubing with compressed air to facilitate the stripping of pipes. The pipes are stripped by two mechanisms constituting a single unit. One mechanism pushes the pipe through 1000 mm partly out of the flask and the other drags the pipe clear of the flask.

The outer surface of the cast pipes is dressed with pneumatic chisels. A grinding stone is used for the internal surface of pipes (Fig. 4). The pipe is mounted on four supporting rollers, and rotary grinding wheels fixed on long shafts, are inserted one in each end of the pipe. The pipe rotates in the opposite direction to the rotation of the grinding stones.

SUBSCRIPTIONS TO INDUSTRIAL, SCIENTIFIC AND TECHNICAL JOURNALS FOR 1959 WILL BE ACCEPTED FROM OCTOBER ONWARDS

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SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosenergoizdat	State Power Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LEIIZhT	Leningrad Power Inst. of Railroad Engineering
LET	Leningrad Elec. Engr. School
LETI	Leningrad Electrotechnical Inst.
LETIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MEP	Ministry of Electrical Industry
MES	Ministry of Electrical Power Plants
MESEP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTI	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NII ZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOE	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIEL	Central Scientific Research Elec. Engr. Lab.
TsNIEL-MES	Central Scientific Research Elec. Engr. Lab.—Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIESKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZEI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us. — Publisher.